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BENEFICIATION OF COMPLEX SULPHIDE ORES FOR ADD-ON PROCESSING

Phase I - Technique Development

by

Don Taylor

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BENEFICIATION OF COMPLEX SULPHIDE ORES FOR ADD-ON PROCESSING

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By: Don Taylor
Date: May, 1989

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EXECUTIVE SUMMARY

The Province of New Brunswick realizes significant economic benefit from its base metal mining operations. For the purpose of encouraging more base metal processing in the Province, this project was undertaken to assess the merits of a centralized custom mill concentrator, capable of supporting add-on processing facilities.

The project is broken down into two phases. Phase I, which forms the substance and scope of this report, deals with the techniques that can be used in the assessment of the custom mill concept. A subsequent phase (Phase II) will propose to utilize these techniques for actual assessment purposes.

Included in this report are the results of a literature study completed on the custom mill concept, including the beneficiation and add-on processing of base metal complex sulphides. As well, various mineral plant simulators were tested, with an emphasis placed on flotation simulation. The report also details the work completed on the design, construction, and operation of a flotation column for application to the beneficiation of complex sulphides.

It is concluded that a custom mill based on a bulk flotation concentrate product has potential for economic gain over current practices due to lower grinding costs and higher overall mineral recoveries. Add-on processing of such a concentrate was shown to have merit, but the technology for such processing still requires further development.

The plant simulators were unable to produce the desired results due to deficiencies in the generation of laboratory test data. Difficulties in processing the Caribou ore prevented determination of the kinetic rate data needed as input to the flotation simulation programs.

Column flotation test work revealed difficulties in maintaining the necessary control over the sparging and liquid levelling requirements. Further laboratory development of the column setup is required before field application.

1.0 INTRODUCTION

1.1 Objective

This is a two-phase project to provide information on the viability of a custom mill for the processing of complex sulphide minerals within the Province of New Brunswick. The custom mill would encourage further in-province base metal processing by exploiting small tonnage mineral deposits. The concept is to construct a centralized base metal concentrator in the "metal belt" of the northern region of the province capable of treating many complex sulphide deposits. The custom mill evaluation will include the possibility of refining the produced concentrates to intermediate metal products.

Phase I of this project is to determine the published background information on this subject and to develop techniques that can be used in the assessment of the custom mill concept.

1.2 Scope

The project (Phase I) has focused on the adaptation and/or development of methods and techniques by which the viability of a custom mill can be evaluated. Activities have included:

1. An extensive literature review of the beneficiation and add-on processing of complex sulphides.
2. The procurement and application of various computer simulation software packages (S.P.O.C., CALFLOT, etc.) including the development of laboratory techniques to provide proper test data for these programs.
3. The construction and commissioning of a column flotation test cell that may be useful in the custom mill flowsheet.

The application of the laboratory and simulation techniques for assessing the custom mill viability will form the substance of Phase II of this project. The second phase is to be recommended for funding provided by the next proposed Mineral Development Agreement.

1.3 Background

The value of metallic mineral production in New Brunswick was $545,957,000 in 1988, which was a significant contribution to the economy of the Province. This production came from Brunswick Mining and Smelting Corporation Limited, Dominion Explorers Inc. (Durham Mines Division), and Gordex Minerals Limited. The valuable base metal concentrates produced were zinc, lead, copper, and antimony; all of which may contain small quantities of precious metals, mainly silver and gold. One producer generates gold bullion from its processing plant. The beneficiated lead concentrate from Brunswick Mining and Smelting Corporation is further refined to lead metal in the company's Belledune refinery. The zinc, copper and antimony concentrates are shipped from the Province for further processing.[1]

There are considerable reserves of metallic minerals (mainly base metals) in the Province with good economic potential. Although some deposits are part of larger mineralized zones, a considerable number are localized deposits[2,3] that are not economical for stand-alone beneficiation plants. Therefore, a centralized processing plant may have some advantage for the exploitation of these smaller mineral deposits.

The base metal deposits of New Brunswick have some unique characteristics that make recovery of the mineral concentrates difficult. Typical base metal plants process mineral grains that average 70 microns in size. The grain size of minerals from New Brunswick deposits is typically 25 to 35 microns, but can be as small as 6 or 7 microns. The fine mineral grain size causes two serious problems:

1. Extremely high grinding costs, as the result of the reduction required for liberation of the valuable minerals from the host rock.
2. The difficulty in recovering the fine-grained minerals in standard metallurgical flotation processes.

This results in typical New Brunswick concentrator plants having high capital and operating costs associated with grinding systems, and poorer than average recoveries of valuable minerals. Therefore, it would seem appropriate for a study of this nature to examine the grind versus grade-recovery relationship for New Brunswick ores. This development may lead to the possibility of producing a "coarser" bulk concentrate that would reflect lower grinding costs. The bulk concentrate production method would produce a poorer grade of mineral product, as the valuable minerals would still have some host rock attachments. However, at the same time, it is expected that there will be higher overall mineral recovery and this, along with the reduced grinding costs, should prove to be of economic benefit.

The reason that current mineral concentrator plants do not produce such bulk concentrates is the difficulty in refining these concentrates to pure metals. Most refineries will only accept high grade concentrates because of the design of the refining process that each plant uses. To process low grade bulk concentrates, these refineries would require major revisions that would greatly increase their refinery operating costs. However, there are new refining processes that are currently in the laboratory or pilot plant stages which could be used on bulk concentrates with reasonable economic efficiency.

To fully evaluate the viability of a custom mill, preliminary economic studies need to be carried out. In an effort to obtain costs for such a plant, all areas from crushing to dewatering need careful consideration. To produce this data reliably, it was determined that the S.P.O.C. programs (Simulated Processing of Ore and Coal) be obtained from CANMET for use in plant design.[4] The computerized programs enable an engineer to design any plant unit system on a computer to determine if the system will work. The size, operation, and type of equipment can be simulated to produce equipment listings that can be combined with appropriate estimation data[5] to produce plant costs.

The most difficult part of the test program is the flotation simulation area, as there are many interacting variables in the flotation operation. The software program (CALFLOT) was obtained from Laval University where it was developed for this purpose. The computer package requires the gathering of data from laboratory flotation tests to produce information on flotation kinetics (the rate at which various minerals float under certain conditions). Once the flotation kinetic rate data is entered into the program, the flotation circuit is then simulated for grades, recoveries, flows, and equipment specifications.

The problem of fine-grained mineral recovery will be examined to determine if a flotation column can improve the overall metallurgical performance. These flotation columns are reported to be very efficient at recovering very fine grain minerals because of the column's flotation action. Therefore, the project will include the construction and commissioning of an in-plant flotation test column.

The base metal sector of the New Brunswick mineral industry accounts for more than 88% of the production value of all metallic minerals produced in the province. Therefore, it is important that research projects be carried out to enhance this sector of the mineral industry. The industry can best be helped by:

1. Improvements in existing technology.
2. The introduction of new technology to current operating plants.
3. Fundamental research into basic mineral processing.

To this end, the Province of New Brunswick and the Government of Canada entered into a Mineral Development Agreement (MDA) for funding such projects. The particular emphasis of this project has been to examine the viability of a custom mill for the Province of New Brunswick. The first phase of this project has been to develop techniques to examine concepts in complex sulphide mineral production and refining. To this end, literature studies were completed, process simulators were obtained and tested, and new technology was evaluated. Further phases of this project will need to be carried out to complete the overall goal of reviewing the viability of a centralized based metal concentrator with the possibility of an add-on processing unit.

2.0 Literature Searches

In an effort to determine the viability of a custom mill for complex sulphide mineral recovery, extensive literature studies were carried out. The first study examined the possible flow schemes to process mined ore to intermediate metal products. A second study was carried out to determine the information available on the various sections of the custom mill operation.

The study on possible flow schemes was compiled under contract by J. McGrath, P. Eng. who wrote the report "Custom Mill Options for New Brunswick Complex Sulphide Ores".[4] The report's recommendations may be summarized as follows:

1. The relationship of grind versus grade-recovery be examined in all areas, especially in grinding economics.
2. To test various reagent applications in a custom mill operation.
3. To test column flotation versus conventional flotation.
4. To test reverse flotation for pyrite removal.
5. A testing program for copper/lead separations.
6. A program to determine if iron can be removed from both concentrates.
7. An add-on extraction unit to process New Brunswick produced bulk concentrates.

The report generally concludes that a custom mill for complex sulphides is a viable concept but requires further evaluation. The report also indicates that the production of a bulk concentrate in the custom mill may be a better way to treat fine-grained complex sulphide ores. The rationale for bulk sulphide production is found in higher mineral recoveries and lower grinding costs.

Extensive literature studies were carried out to determine the most recent information available on the various sections of a custom mill. The literature search was made in the following sections, and a complete listing of all papers obtained is given in Appendix I.

1. Copper metallurgical operators
2. Zinc metallurgical operators
3. Lead metallurgical operators
4. Simulation
5. Mineralogy
6. Crushing and grinding
7. Standard flotation
8. Column flotation

Considerable emphasis was placed on the production of bulk sulphide concentrate in the custom mill proposal. The production of bulk sulphide would require an examination of particle size and mineralogy as related to the liberation of complex sulphide ores. In the literature search for information on this subject, it was found that many studies had been completed on New Brunswick ores.[7-17] From the literature study, it was apparent that fine grinding was needed for these complex sulphide ores. The literature available indicated that the liberation size for Brunswick Mining and Smelting ore was about 37 microns, and for Caribou ore it was 9 to 16 microns. With this available literature on particle size analysis, it was determined that more liberation analysis work under this project was not necessary.

The literature searches were accomplished using standard computer search techniques, and the Datapack and Dialog computer information systems. The databanks used in the literature searches were "Mineral Processing Abstracts" from CANMET and "Compendix Plus" from Engineering Information Inc.

3.0 Test Work Rationale

From the recommendations of the report on "Custom Mill Options", a test program was developed to provide the necessary data for assessment of the custom mill concept for base-metal processing. The program is summarized as follows:

1. The use of plant simulators for crushing, grinding, and dewatering to determine the size, type, and cost of equipment for a complex sulphide mill producing bulk concentrates.
2. Flotation simulators would be tested using laboratory data obtained from bench-scale tests.
3. The testing of flotation columns versus standard plant flotation machines should be done by operating a small-scale research flotation column system in a base metal flotation plant. The data from the tests would show if flotation columns would be more effective in recovering fine-grained minerals.

3.1 Plant Simulators

In recent years, extensive work has been carried out on plant simulators to aid mineral processing plant personnel to better understand and improve their operations. One of the best known simulation systems is the S.P.O.C. (Simulated Processing of Ore and Coal) system, which was designed and engineered by CANMET.

The programs for some of their simulators were obtained from the agency, complete with descriptive information of their operation. These programs were initially run on a C.D.C. Cyber 730 computer, and were later converted for use on P.C. compatible machines. All examples of data runs given in the manuals were taken from the C.D.C. computer system. Therefore, it was decided to carry out the following taskings as a way of adapting these simulators:

1. The P.C. computer that was available for use required the following modifications in order to run the S.P.O.C. simulator programs.
    a. A Microsoft Fortran Compiler was installed, as the S.P.O.C. programs are written in Fortran, not in P.C. Basic language.
    b. A high speed 8087 mathematics co-processor module was added to the computer's C.P.U. in order to handle the large mathematical calculations required in the simulator.
2. The S.P.O.C. programs were first tested with their operating manuals' "test" data to verify the program's correct operation.
3. Known plant configurations (Brunswick Mining and Smelting and Dominion Explorers Inc.) were put into the system to determine if the simulator would produce a reasonable simulation of the plant data.
4. To incorporate a plant design as input to the simulators and to obtain the simulation data for a custom mill, which would be used for economic evaluation of the project.

The first S.P.O.C. system to be evaluated was the crusher simulator. The simulator was first operated with the operating manual's "test data". The input of data to the crusher program was found to be complicated and several days were needed before the "test data" could be properly input. Once the data input had been successfully made, the program ran well and produced the required information (see Appendix 2).

Two runs were completed to test the crusher simulator against actual plant applications. Both runs gave poor simulation results due to the unusual screen designs in each plant (See Appendices 3 and 4). It appears that the simulator is restricted to very standard screening operations and, therefore, the simulator must be used with standard equipment configurations.

Due to the extensive work needed to put the crushing simulator into operation, and the extreme complexity of the grinding simulator, it was determined that work on the grinding system would not be started until the flotation simulator was operational. Once work had been completed on the crushing and grinding simulators, a test of the dewatering simulator could then be carried out.

3.2 Flotation Simulation

The flotation simulator available in the S.P.O.C. system could only work on data from an existing plant. As a result of this limitation, a better flotation simulator was searched for and obtained from Laval University in Quebec City. This program, CALFLOT, was able to simulate flotation circuits by the use of flotation rate kinetics, and data on flotation machines that had been supplied by manufacturers. The simulator is designed to produce the configuration of flotation flows, the number of cell banks, the number of unit cells per bank, the volume of each cell, the types of cells, and other engineering data. The simulator also produces flow rates of concentrates, tailings, water, and a complete cell by cell profile of grades and distribution of flotation products.

After receiving the computer program, the following test objectives were:

1. To test the program on the operating manuals "test" data to prove the system would operate on the available P.C. machine.
2. To obtain flotation rate data in the laboratory from actual operating plant feed. This would be used to verify that the simulator could accurately simulate the actual plant's operation from the flotation rate kinetic data generated by testing.
3. To obtain flotation rate kinetic data in the laboratory for complex sulphide ores, and then simulate a possible custom mill flotation circuit that could be used in the design of such a mill.

The flotation simulator was obtained and installed on a computer. This simulator was found to be user friendly and worked well on the supplied data. However, problems began to appear when we tried to generate our own kinetic rate parameters for the simulator. Test work was carried out on an antimony ore to verify the correct operation of the simulator. The laboratory test work was found to be tedious, time consuming, and required patience and extremely careful work. When the tests were completed, it was necessary to calculate the kinetic rate parameters. However, this proved to be much more difficult than originally envisioned.

After several consultations with Laval University, a mathematical-graphical method was found useful for determining the kinetic rate parameters. This method, along with a similar method using cell concentrations, required interpolation of graphical data. Depending on how the "lines" were drawn through the "scattered points", great variances in rates could occur.

In an attempt to make more accurate results available, it was decided to examine computer statistical packages to help us interpret the kinetic rate data. When we inquired if Laval University could help us with the use of computer statistics, it was learned that they were in the process of developing such a package for their kinetic rate data tests. Therefore, it was decided to obtain this new program called CALIFLOT.

A verification of the procedures to calculate kinetic rate data was then carried out, confirming that the three methods of calculation would produce the required data with reasonable accuracy (see Appendix 5). The use of CALFLOT and CALIFLOT were written in a report to show the operating procedures and results of the flotation unit simulator (see Appendix 6).

The simulator was then tested on the antimony ore from Dominion Explorers Inc. at Lake George, New Brunswick to confirm that this simulator could detect variances in kinetic rates. A series of laboratory tests were carried out to vary the kinetic rate by changing the quantity of collector used in each test. The test work confirmed that the simulator could easily detect these kinetic rate changes (see Appendix 7). With the results of test simulation studies and previous work on flotation of Caribou ore (see Appendix 8), it was determined that flotation tests for kinetic rate data should be carried out on the Caribou complex sulphide ore.

In order to ensure proper plant simulation, good laboratory flotation test data is essential. The laboratory test work must be able to simulate the best results obtainable in the producing plant. However, complex sulphide ores are one of the most difficult ores to float in the laboratory and even in a plant environment. This is mainly due to the fine grinding necessary to liberate the useful minerals from the host rock and the difficulty in floating such fine minerals.

The test work on Caribou ore turned out to be more difficult than anticipated. The laboratory program to produce a proper laboratory flotation procedure ended without the completion of a proper kinetic rate flotation test. The laboratory testing involved six fractional factorial design tests, encompassing a total of 24 tests. Variations in reagent types, reagent feed rates, pH, and depressants were tested without much success. Consultation was carried out with East West Caribou Resources to improve the laboratory procedures, but poor results continued (see Appendices 9, 10, 11, 12, 13, 14).

3.3 Flotation Columns

The standard flotation cell consists of a mechanically driven agitator in a rectangular tank, with low pressure air being forced into the mixing zone of the ore-pulp solution. These cells tend to be very efficient for the flotation of normal concentrate feed sizes. However, they tend to have poor performances in the very fine size fractions (under 20 microns). A different method of froth flotation is now being used in an attempt to float fine concentrate size fractions efficiently. This flotation system has been in research and development for the past 20 years and is called column flotation.

In these column flotation units, there is no mechanical agitation since the units use air sparged in at the bottom for mixing, as well as for the flotation operation. These units tend to be slender and tall (commercial units being a metre in diameter and 10 metres tall). Several commercial units are in operation and have shown that fines flotation in a column can be very efficient. With this information on column flotation, the following program was designed:

1. To complete a literature search on laboratory column flotation units and obtain all data possible on commercial units.
2. To select a design and build a laboratory flotation column to enable test work to be carried out.
3. To take the column to an operating plant and test the unit against standard flotation machines. This would be used to verify that the column unit could produce better grades and recoveries, especially in the fine size fractions.
4. To obtain the necessary information to be able to scale laboratory columns to plant size equipment so the costing of commercial units would be possible.

An extensive literature search was carried out on flotation columns to determine laboratory and industrial use of these units, and the procedures needed to scale-up from laboratory to industrial columns. The papers studied for the flotation column systems are shown in Appendix 15.

A two-inch test column was constructed based on the design supplied by the United States Bureau of Mines[18] who have completed extensive work on column flotation. The column was set up in the laboratory and tested with water. The column appeared to work well until it was exposed to plant flotation feed. The column developed problems with feed and tail ports sanding up, with level control, and with the air sparging system. Test work was attempted on some grade-recovery relationships for the antimony ore at Dominion Explorers Inc. and showed limited success. From test work completed on the column to date, it was clear that further work is necessary with respect to the air sparge system and the operating procedures (see Appendices 16, 17, 18, 19, 20, 21).

In conjunction with these tests, the method for column scale-up was investigated. A report was prepared outlining a procedure by Dobby and Finch, which can be used to scale-up columns (see Appendix 22).

4.0 Conclusions

Phase I of this project has provided a good basis for the information and technology needed for Phase II of this proposition. The literature studies completed indicate that a bulk sulphide custom mill is a viable unit, but the subsequent processing of the concentrates in existing refineries is not possible. Therefore, the custom mill must produce high grade concentrates or have its own add-on processing system to convert concentrates to intermediate metals. The economic study for a custom mill will need to be carried on in the second phase of this project.

The plant simulators for crushing and flotation are able to provide the necessary information to make preliminary cost estimates. The determination of kinetic rate data for the flotation simulator, however, is a very difficult process. Considerable amounts of test work to produce the needed kinetic rate data for complex sulphide ores will be needed in Phase II of this project.

The grinding and dewatering simulators have not been tested, therefore, evaluation of these units will need to be completed in the next phase of this project.

The flotation column would not operate properly due to design and engineering problems. All effort must be put into perfecting the operation of this unit in Phase II of this project. It is apparent that fine grinding will be needed even in a bulk sulphide concentrate process, and the use of columns will be very important in a custom mill.

5.0 Recommendations

5.1 Plant Simulators

The plant simulators available from S.P.O.C. need modification to enable easier input of data along with more parameter adjustments in various operating units. The S.P.O.C. programs have been completely modified by CANMET and are now available in a user friendly input designed for P.C. type computers. Therefore, it is recommended to purchase these modified programs, and to test them with actual plant data in Phase II of the custom mill project.

5.2 Flotation Simulation

Considerable work is needed to determine acceptable flowsheets for each complex sulphide ore. A program to work one of these ores will require many laboratory tests and careful engineering work. Once the flowsheet has been determined, the kinetic rate tests can be completed. However, considerable time and effort will need to be appropriated in Phase II of this project to complete the flowsheets for complex sulphide ores.

The kinetic rate tests are now conducted in a standard laboratory flotation machine. The procedures used for this test are very long and complicated, with room for many errors. In an effort to reduce errors, a new style of flotation machine is available from Laval University. This flotation machine is especially designed for use on flotation rate tests. The machine has automatic cell level control, special froth removal systems, computerized power and air flow measuring devices, and other features for kinetic rate studies. Therefore, the purchase of this new flotation machine should be considered before work on Phase II of the custom mill project is started.

5.3 Flotation Column

The flotation column would not operate properly during the test work that was attempted. In consulting with the University of Toronto Metallurgical Department, they advised that it took them five to seven years to get their test columns operating properly, using engineering students, graduate study students, and their academic staff. The following program, is recommended to be used by Phase II of this project:

1. Placing as much automatic control as possible on the column; including level control, air supply, water supply, and other important parameters.

2. Taking our column to our laboratory and working with it using recirculating slurries. One major problem here is the height of the column (laboratory ceilings are eight feet high and a column requires 30 foot high ceilings). Therefore a satisfactory place for column erection and operation is necessary.

3. To send an engineer and/or technician to the University of Toronto for a two week period to learn all they can about the operation of columns in their laboratory and to determine their experience in plant testing of columns.

4. Extensive work is needed on the air sparging system of the column as this is the key to proper fines flotation. Many types of systems have been tested, but they have not produced a good stable air flow in the column.

Flotation column test work will need to be continued in Phase II of this project, as it is expected that flotation columns will perform a major role in mineral recovery in the custom mill plant.

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APPENDIX 1

Process Engineering Develoment Papers

COPPER PAPERS

PAPER NUMBER	AUTHOR	TITLE	SOURCE
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APPENDIX 2

Crushing Plant Simulator

PROCESS STUDIES GROUP

technical notes

Project No.		*File No.*	
Project Plant Simulators		*Date* 17/07/88	*Page* 1

***Subject*:** Crushing Plant Simulator

***Author(s)*:** Donald S. Taylor

***Objective*:** To determine the operating data and parameters to successfully run the CRSHEX simulator program on the Department's micro computers.

***Conclusions*:**

The CRSHEX simulator program was successfully run on the data from the CRSHEX operating manual, and data from operating crushing plants. The program works reasonably well, but is not user-friendly. Therefore, operating the program requires careful planning and close attention to all details in order to complete an executable crushing circuit.

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Discussion: The CRSHEX simulator is a unit of the CANMET programs put out under the S.P.O.C. system (Simulated Processing of Ore and Coal). This program cannot be run as a normal S.P.O.C. program because it requires its own special disk boot system. The system for using CRSHEX is as follows: when at C-prompt (C:>), insert the disk in "A" drive and push the CTRL, ALT, DEL keys together. This causes the system to re-boot, and when the computer looks for its operating system, it looks on "A" drive first, thus, booting up the special "A" drive boot operating system.

When this is completed and the A-prompt (A:>) appears, type in "CRSHEX" (which starts the program). The program then asks for name, reference number, and date (Figure 1). Next, the program asks for the type of, condition, and density of the material to be crushed. The Bond Index, % humidity, and process tonnage are also needed (Figure 1).

Next, the program asks for a feed size distribution, where a feed size can be chosen from their recommendations, or if one has the screen analysis of the feed, it can be entered in under "other" (Figure 2). The program will then request the "Unit Specifications" of which you are given seven choices (Figure 2).

The first unit you must specify is "Feed" so the selection will be "7". The program must always start with "Feed" and end with "Stock Pile-Final". If these units are not there, the program will not work. Both of the above units will be discussed in the listing below.

The next unit is specified by choosing the equipment in Figure 2. When a unit is specified, the following information is needed:

1. Crusher: choose among jaw, gyratory, cone standard, cone short head, roller, impactor, hammer, or "other" crusher units (Figure 3). For the jaw, gyratory, or cone crushers, the following information is needed - model, capacity, maximum opening, and setting (Figure 3). The roller crusher requires the above information plus the roll diameter, the roll length, the rotary speed, and an estimate of the roller product size (Figure 4). The impact crusher and crusher hammer require the model, capacity, maximum opening, screen opening, and rotary speed (Figure 4). The "other" crusher designation for crushers requires a model, capacity, maximum opening, and a setting for the unit (Figure 5).

2. Screen: the screen model requires the following information. It first must know if the deck is horizontal or inclined, and the number of decks on the screen (Figure 6). The program then asks for the model, angle, water flow, and number of sections. For each section the following information is needed: length, width, wire opening, and wire diameter (Figure 6).

Discussion: 3. Separation: the separation only needs the number of streams required, since it divides the feed equally among each stream (Figure 7).

4. Recombination: recombination is used to bring two or more streams back together. There are two modes for recombination; one a normal mode where two streams mix, and the second mode has recirculation where one piece of equipment has a circulatory load around it (closed loop on a secondary crusher). The program will ask for the number of recombining streams from 2 to 4 (Figure 7).

5. Silo: silo is used for temporary storage during crushing operations. The silo system requires a capacity and flow rate for the model (Figure 8).

6. Stock Pile: stock pile operates in two modes; buffer or final. The final mode is the end storage bin for the crushing system. The buffer stock pile is for temporary storage, and requires a flow rate in order to complete the program (Figure 8).

7. Feed: the feed system operates with or without a grizzly feeder. If there is no grizzly, the program just continues. If there is a grizzly, the program requires the type of feeder, length of feeder, width, length of grizzly, width of bars, opening, and angle of grizzly (Figure 9).

When all of the equipment has been specified, the input section of the program needs to be ended. Before doing this, it is essential to:

1. have a "feed".
2. have a "stock pile final".
3. determine if all of the equipment needed is in the program.

Once you proceed past this point, it is difficult to add and impossible to remove information, and in some systems, it is not possible to update information in the program. The only way to solve any of the problems above is to "end" the CRSHEX program, and start all over again.

After the last piece of equipment is specified, push "return" and a grid system appears on the monitor, this is the beginning of putting the equipment together in logical strings. The bottom line under the grid system is the most important one since it asks for unit to unit specifications. It is essential to have a flow diagram with unit numbers on each unit to determine flows between units (Figures 10 and 11).

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Method: When the last unit is put into the crushing logic strings, the computer prints "Start over from which stream? [ ] <ret> continue" on the monitor, then you just push the "return" and the computer goes into the execution mode (Figure 12). The monitor displays "Execution of Model" then asks "Do you wish to modify certain parameters of the model simulation (Y/N)?". If you tell the computer no (N), it goes on to execute, if you enter yes (Y), you will need to know modification numbers, percentage feed settings, and other information (Figure 13) in order for the program to execute.

At the end of execution, the computer puts up boxes to show the connection of the first few pieces of equipment. After the boxes are in place, the computer asks for D)escription (a description of the unit) (Figure 14), G)ranulomytrie (the size distributions of the streams on the computer screen) (Figure 15), and U)nit (which asks for the next unit to be seen). However, that unit must be on the screen or you must move through the units until the desired unit is reached. This means that you cannot move from unit 1 to unit 6 without going through units two to five.

There is no print mode in the program itself, however, "print screen" can be utilized to print what is on the monitor as you need the data. The procedure is "shift" and "Prt Scn" (at the same time) to print the contents of the screen on the printer.

It is possible to modify parameters in some of the unit modules, crushers and screens are easy to modify. In order to modify this equipment, we need to proceed to the "Unit Specifications" screen, and then type in "U" for update, and then the unit number for the screen equipment to be updated. When the unit appears on the screen, type in the unit specification number "3", then the new parameter (Figures 16 and 17).

FIGURE 1

CRUSHER SIMULATION PACKAGE

USER IDENTIFICATION

1. Name: [ ]

2. Reference number:

3. Date:

Feed data

1. Type of Material : [ ] 1) Limestone, 2) Granite, 3) Trap Rock
4) Ore, 5) Others

2. Condition of Material:

3. Density of Material : tonnes / cubic metre

4. Bond Index (WI) : Kwh / tonne

5. % of Humidity (H2O) %

6. Tph (metric) to process : tonne / hour

FIGURE 2

```
                 ┌──────────────────────────┐
                 │  FEED SIZE DISTRIBUTION  │
                 └──────────────────────────┘

           Please enter the FEED size distribution


 1. Granite    0-36 in              9. Trap Rock   0-36 in
 2.  "  "      0-24 in             10.   "    "    0-24 in
 3.  "  "      0-20 in             11.   "    "    0-20 in
 4.  "  "      0-18 in             12.   "    "    0-18 in
 5. Limestone  0-36 in             13. Gravel      0-12 in
 6.   "    "   0-24 in             14.   "  "      0-8  in
 7.   "    "   0-20 in
 8.   "    "   0-18 in             15. Other



              SELECTION [   ]
```

```
                    UNIT SPECIFICATIONS

                    UNIT NO   .


    1. Crusher                          5. Silo

    2. Screen                           6. Stock Pile

    3. Separation                       7. Feed

    4. Recombination



SELECTION:     <ret>continue  - (#)unit type
               E)nd (stop)
```

FIGURE 3

```
                    UNIT SPECIFICATIONS

                 UNIT NO   . - CRUSHER -

      1. Jaw                            5. Roller

      2. Giratory                       6. Impact

      3. Cone ST.  .                    7. Hammer

      4. Cone SH.                       8. Others




  SELECTION:    C)ancel
```

```
                    UNIT SPECIFICATIONS

                 UNIT NO   . - CRUSHER    JAW

  1. Model:  [                    ]

  2. Capacity:         tonne

  3. Maximum opening:           mm

  4. Setting:       mm
```

FIGURE 4

UNIT SPECIFICATIONS

UNIT NO . - CRUSHER ROLLER

1. Model: [ ]

2. Capacity: tonne

3. Maximum opening: mm

4. Setting: mm

5. Roll diameter: mm
6. Roll length: mm
7. Rotary speed: RPM

UNIT SPECIFICATIONS

UNIT NO . - CRUSHER IMPACT

1. Model: [ ]

2. Capacity: tonne

3. Maximum opening: mm

4. Screen opening: mm

5. Rotary speed: RPM

FIGURE 5

UNIT SPECIFICATIONS

UNIT NO . - CRUSHER Type Undefined

1. Model: [ ]
2. Capacity: tonne
3. Maximum opening: mm
4. Setting: mm

FIGURE 6

```
                              UNIT SPECIFICATIONS

                             UNIT NO    . - SCREEN -

        HORIZONTAL -                            INCLINE -
                      1. One deck                            5. One deck
                      2. Two decks                           6. Two decks

                      3. Three decks                         7. Three decks

                      4. Four decks                          8. Four decks




        SELECTION:    C)ancel
```

```
                              UNIT SPECIFICATIONS

                             UNIT NO    . - SCREEN/INCLINED, 1 DECK

1. Model:                     Deck no.1 3. Water     0 litre  per minute

2. Angle :   0          )             4. Numb of sections:  1

                           SECTION 1

 Length:               5.[     ]mm

 Width:                6.      mm

 Wire opening:         7.      mm

 Wire diameter:        8.         mm
  (0 = slotted)
```

FIGURE 7

```
                     UNIT SPECIFICATIONS

                  UNIT NO   . - SEPARATION -

Number of streams: [ ]   ( 2, 3, 4 or C)ancel  )
```

```
                     UNIT SPECIFICATIONS

                  UNIT NO   . - RECOMBINATION -

                  N)ormal

                  R)ecirculation



SELECTION:    C)ancel
```

```
                     UNIT SPECIFICATIONS

                  UNIT NO   . - RECOMBINATION -


                  R)ecirculation


Number of streams: [ ]   ( 2, 3 or 4 )
```

FIGURE 8

UNIT SPECIFICATIONS

UNIT NO . - SILO -

Capacity:[ ] tonne

Flowra tonne /hr,

Time to attain maximum capacity: hr

UNIT SPECIFICATIONS

UNIT NO . - STOCKPILING -

B)uffer

F)inal

UNIT SPECIFICATIONS

UNIT NO . - STOCKPILING -

B)uffer

Flowra[ ] tonne /hr,

FIGURE 9

```
                         UNIT SPECIFICATIONS

                        UNIT NO   . - FEEDER -

                        G)RIZZLY

                        W)ithout GRIZZLY




SELECTION:    C)ancel
```

```
                         UNIT SPECIFICATIONS

                        UNIT NO   . - FEEDER - WITH GRIZZLY

      1. Type: [                                         ]

      2. Length of feeder:                 mm

      3. Width of feeder:                  mm

      4. Length of grizzly:          mm

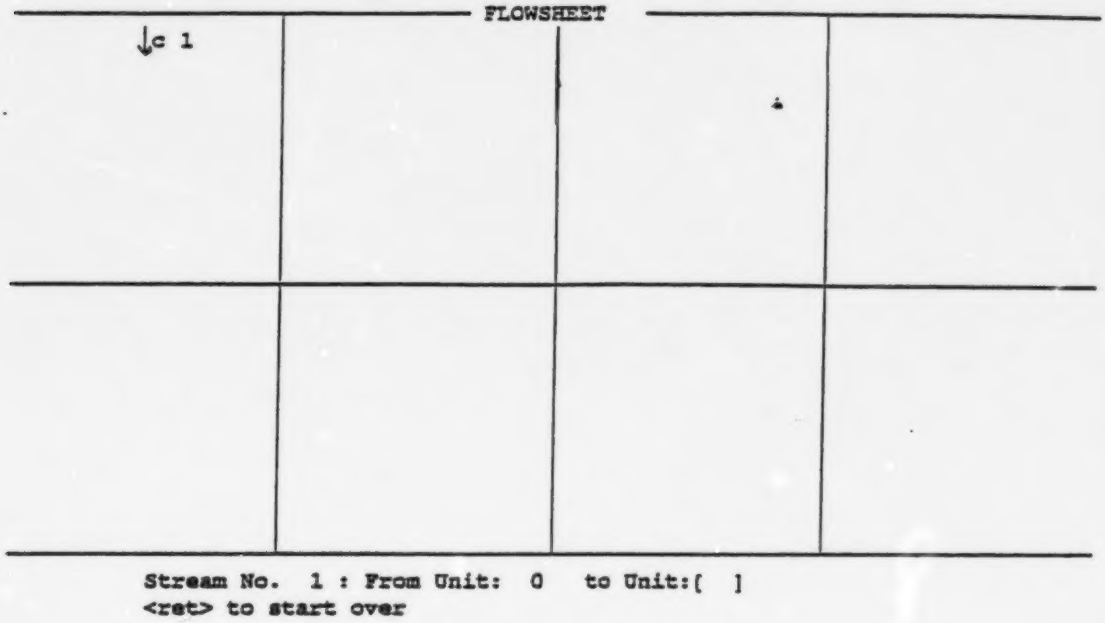
      5. Width of bars:             mm

      6. Opening:         mm

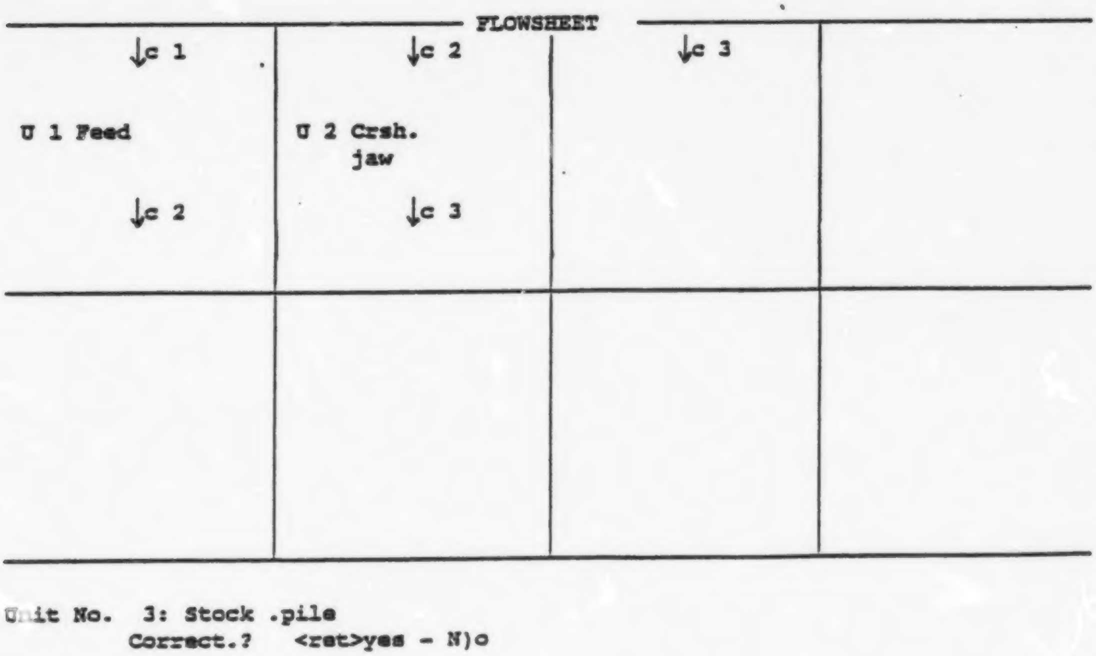
      7. Angle (0 - 30):

         Efficiency:        %
```

FIGURE 10


FLOWSHEET
↓c 1

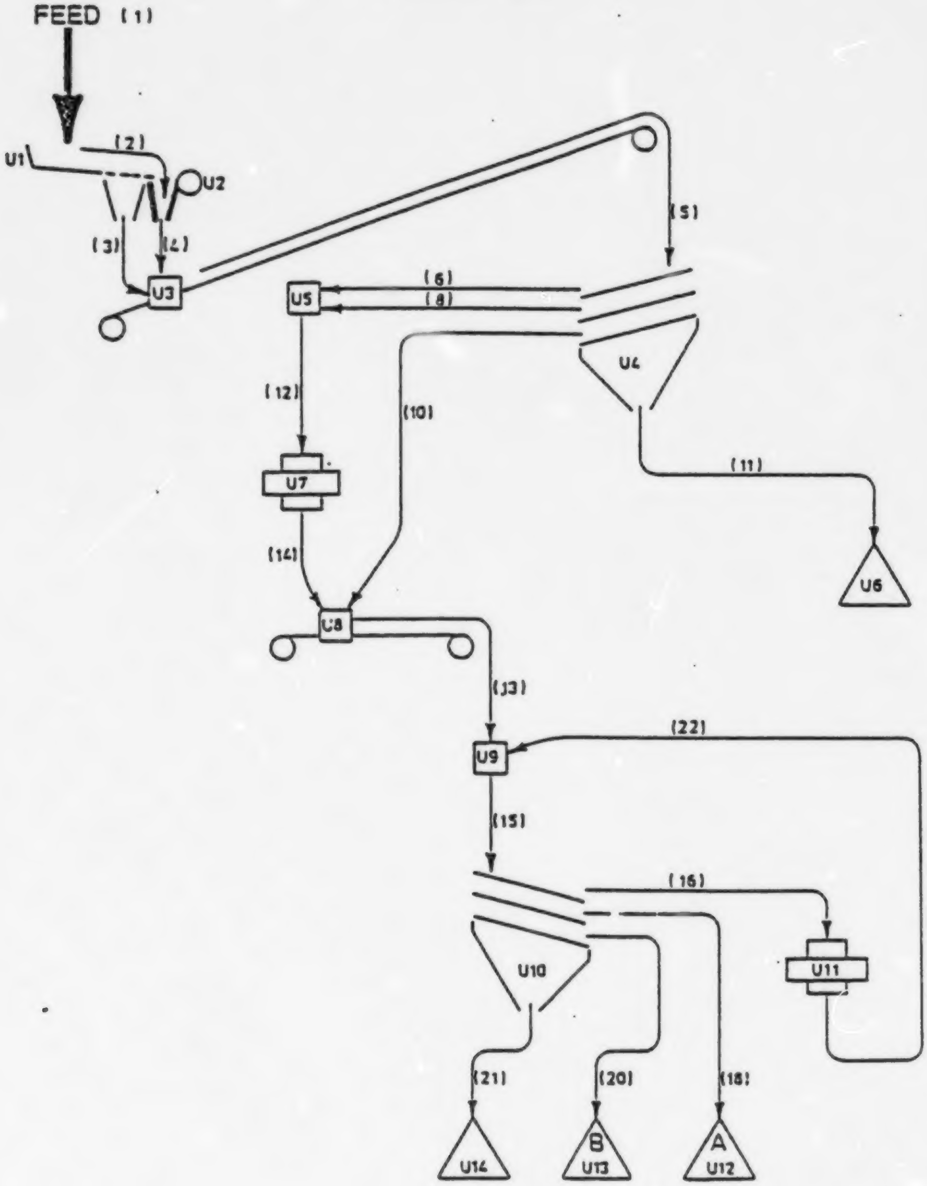
```
Stream No.  1 : From Unit:  0   to Unit:[  ]
<ret> to start over
```


FLOWSHEET
↓c 1
↓c 2
↓c 3
U 1 Feed
U 2 Crsh.
jaw
↓c 2
↓c 3

```
Unit No.  3: Stock .pile
       Correct.?  <ret>yes - N)o
```

FIGURE 11

FLOW SHEET


FEED (1)
U1
(2)
U2
(3)
(4)
U3
(5)
(6)
(8)
U5
U4
(12)
(10)
(11)
U7
(14)
U6
U8
(13)
(22)
U9
(15)
(16)
U10
U11
(21)
(20)
(18)
U14
B
U13
A
U12

Proposed flowsheet for producing products A and B

FIGURE 12

```
-------------------------- FLOWSHEET -----------------------------------
       ↓c 1          |        ↓c 2        |       ↓c 3        |
                     |                    |                   |
  U 1 Feed           | U 2 Crsh.          | U 3 Stock         |
                     |       jaw          |       Pile        |
                     |                    |                   |
       ↓c 2          |        ↓c 3        |                   |
                     |                    |                   |
------------------------------------------------------------------------
                     |                    |                   |
                     |                    |                   |
                     |                    |                   |
                     |                    |                   |
                     |                    |                   |
------------------------------------------------------------------------

          Start over from which stream?        [ ]   <ret>continue
```

```
                    EXECUTION OF MODEL
                    ------------------

Do you wish to modify certain parameters of the model
simulation (Y/N)?
```

FIGURE 13

```
                    EXECUTION OF MODEL
                    ------------------

                      CRUSHER



 What percentage of feed setting would you like ?
1- 70%, 2- 75%, 3- 80%, 4- 82%, 5- 85%, 6- 88%, 7 - 90%[ ]
```

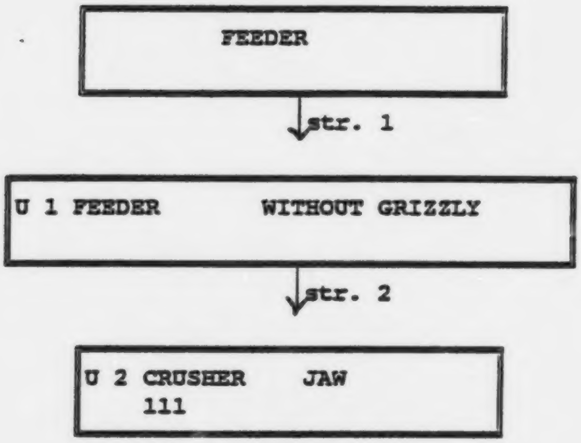
```
                    EXECUTION OF MODEL
                    ------------------

                      CRUSHER



 Modification to feed percentage    , crusher setting
Give new value for modification puposes     ( 1% - 100%): [     ]
```

FIGURE 14


FEEDER
str. 1
U 1 FEEDER WITHOUT GRIZZLY
str. 2
U 2 CRUSHER JAW
111

```
D)escription - G)ranulomytrie - U)nit  - <ret>continue  [ ]
```

```
                    DESCRIPTION of unit no.     2

                                    - CRUSHER    JAW


        1. Model:   111

        2. Capacity:    234  tonnes

        3. Maximum opening:       914 mm

        4. Setting:    45 mm

           Efficiency :  100.0 %
           R/Reduction:   14.7
           Energy consumption:        53.1 kwh


G)ranulomytrie - <ret>continue  [ ]
```

FIGURE 15

```
                     str.  2          str.  3
610    mm            90.0 %           100.0 %
305    mm            56.0 %           100.0 %
203    mm            42.0 %           100.0 %
152    mm            34.0 %           100.0 %
102    mm            25.0 %           100.0 %
 89    mm            22.0 %           100.0 %
 76    mm            18.0 %           100.0 %
 63    mm            14.0 %            98.9 %
 51    mm            11.0 %            94.1 %
 38    mm             8.0 %            82.5 %
 25    mm             5.0 %            59.8 %
 19    mm             3.0 %            48.1 %
 12.7 mm              2.0 %            34.3 %
  9.5 mm              1.0 %            27.4 %
  4 mesh               .3 %            17.6 %
  8 mesh               .0 %            12.2 %
 16 mesh               .0 %             9.0 %
 30 mesh               .0 %             6.9 %
 50 mesh               .0 %             5.4 %
200 mesh               .0 %             3.9 %
  Tph                98  tonnes        98  tonnes
  Water               2  tonnes         2  tonnes
Gr. max.            914 mm             65 mm
 D)escription - <ret> continue :
```

FIGURE 16

```
                    UNIT SPECIFICATIONS

                       UNIT NO   .

      1. Crusher                         5. Silo

      2. Screen                          6. Stock Pile

      3. Separation                      7. Feed

      4. Recombination




SELECTION:    <ret>continue   - (#)unit type
              U)pdate         - E)nd          - L)ist
```

```
                   UPDATE UNITS

                UNIT NO:   [ ] <ret>continue
```

```
                   UPDATE UNITS

                UNIT NO:.    - CRUSHER    JAW

1. Model:   111

2. Capacity:    234  tonnes

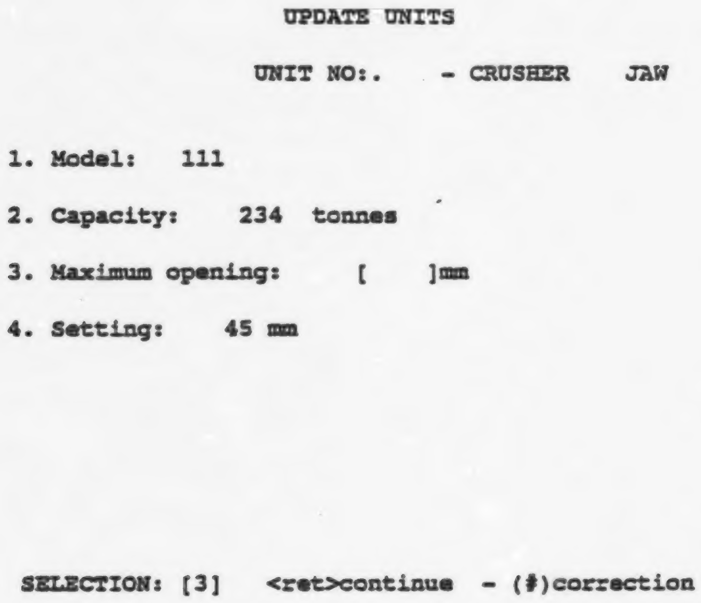
3. Maximum opening:        914 mm

4. Setting:    45 mm



SELECTION: [ ]   <ret>continue  - (#)correction
```

FIGURE 17



```
                    UPDATE UNITS

                 UNIT NO:.    - CRUSHER    JAW

1. Model:   111

2. Capacity:    234  tonnes

3. Maximum opening:     [     ]mm

4. Setting:    45 mm




 SELECTION: [3]   <ret>continue  - (#)correction
```

APPENDIX 3

CRSHEX Simulator Verification

PROCESS STUDIES GROUP

technical notes	*Project No.*	*File No.*
Project Plant Simulator	*Date* 08-07-88	*Page* 1

Subject: CRSHEX Simulator Verification

Author(s): Donald S. Taylor

Objective: In the past few years, simulators have become available from CANMET to look at unit processes in metallurgical plants. The Department of Natural Resources and Energy has obtained these simulators for use in determining improvements in current plant operating conditions and for simulation of data for new mineral processing plants. In order to prove these simulators as a useful tool, it was determined that these unit simulators be tested on processing units already operating in the province.

The first simulator tested was the crushing simulator "CRSHEX", obtained from CANMET. Dominion Explorers Inc. at Lake George agreed to provide the necessary data for the simulation verification on their plant crushing circuit.

Conclusions: The data from Dominion Explorers Inc. crusher circuit was run on the simulator. The results of the final product from the simulator were different from the crushing plant operating data. The screen analysis for the final plant product and CRSHEX simulation program are shown in Figure 1. The graph shows a considerable error in percent passing size analysis in the +5,000 micron size range.

The major reasons for these errors are: (1) the slabby nature of the ore being crushed, and (2) the slotted bottom screen deck on the plant screen. The bottom deck is not a conventional square deck design (½" x ½" squares), but a slotted design, where the holes are ½" x 3", forming a rectangular screen hole. It is very probable that this rectangular screen design would let through larger rocks (especially from slabby ore) than a conventional square screen deck.

The screen model in the CRSHEX program does not appear to take into account slotted screen parameters except in capacity calculations. This can be confirmed by examining the graph above 12,700 microns (½") where no material should get into the product bigger than 12,700 microns due to the bottom deck screen size. The plant data shows 80% passing 12,700 microns, which is what should be expected for a normal 12,700 micron screen. It appears that there is a lot of coarse rock going through the bottom screen.

Author(s)	technical notes	Project No.	Page	of
D. Taylor			2	3

Discussion: The simulator package appears to work well, but is somewhat restricted when nonstandard operating parameters are used. In checking the screen model, it was found that the Karrah model for screen size analysis was used. This model does not appear to take into account the use of slotted screens. Any modification to the program to correct this problem could only be carried out by CANMET at our own expense.

Further testing of this simulator will be carried out on the Brunswick Mining and Smelting Ltd. crusher plant, and published in a Technical Report.

The size analysis of each equipment stream and the equipment specifications from the CRSHEX simulator are shown in Figures 2 to 12. The screen analysis of the actual plant feed is shown in Figure 13.

Author(s)	technical notes	Project No.	Page of
D. Taylor			2 3

Method: An ore sample was collected from Dominion Explorers Inc. on May 17, 1988 along with a description of all plant crushing equipment. The ore sample was dried and run on a standard screen analysis in the research laboratory and reported in Figure 13.

A drawing of the circuit was obtained (Figure 3), and along with the data obtained, the information was placed in the CRSHEX simulator. The feed size specifications for CRSHEX were given as -18 inch trap rock, as the underground operation at the mine produces -18 inch material via its dump pocket, grizzly, skip operation.

FIGURE 1

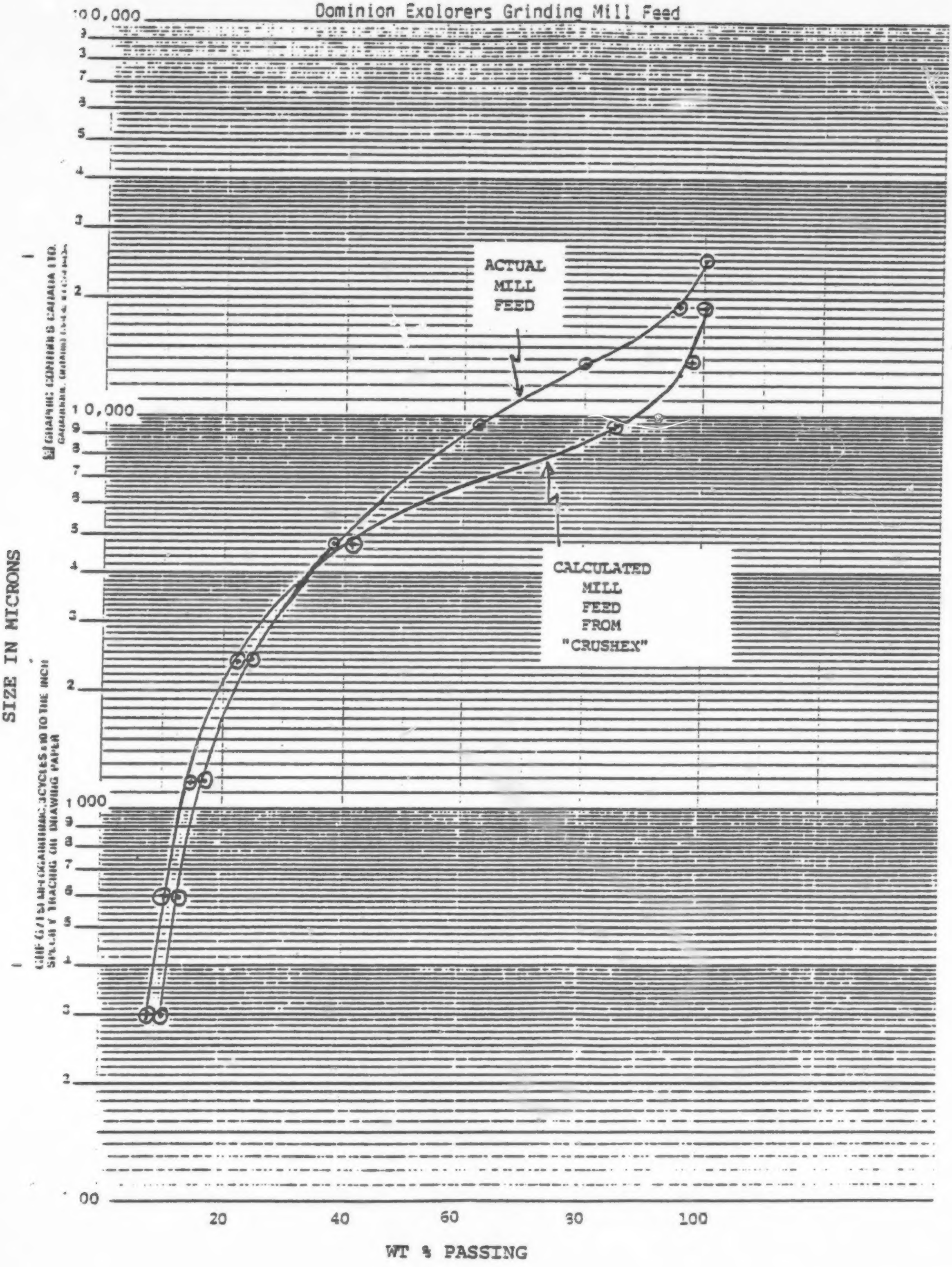
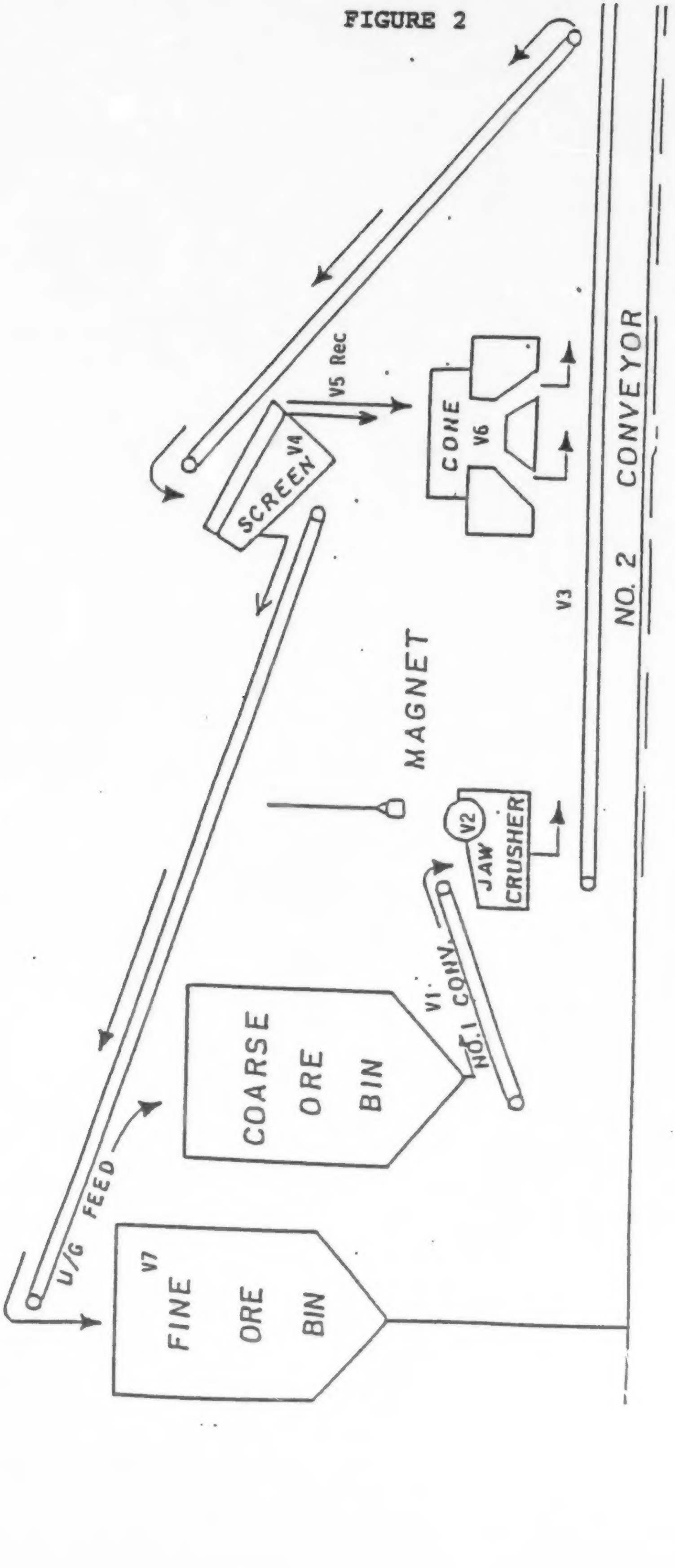

Dominion Explorers Grinding Mill Feed
100,000
10,000
1 000
00
SIZE IN MICRONS
WT % PASSING
20
40
60
30
100
ACTUAL
MILL
FEED
CALCULATED
MILL
FEED
FROM
"CRUSHEX"
GRAPHIC CONTROLS CANADA LTD.
SPECIFY TRACING OR DRAWING PAPER

FIGURE 2


FINE
ORE
BIN
V7
U/G FEED
COARSE
ORE
BIN
NO. 1 CONV.
V1
JAW
CRUSHER
V2
MAGNET
SCREEN
V4
V5 Rec
CONE
V6
V3
NO. 2 CONVEYOR

DOMINION EXPLORERS INC.

FIGURE 3

CRUSHER SIMULATION PACKAGE

```
                    USER IDENTIFICATION

                      1. Name:  DOMINION EXPLORERS

                      2. Reference number:   1

                      3. Date:  17/ 5/88




   SELECTION:    <ret>continue  - (#)correction - E)nd simulation




                         ┌─────────────────┐
                         │    Feed data    │
                         └─────────────────┘

1. Type of Material     :  Ore

2. Condition of Material:  WET

3. Density of Material  :    2.700    tonnes / cubic metre

4. Bond Index (WI)  :       11.270  Kwh / tonne

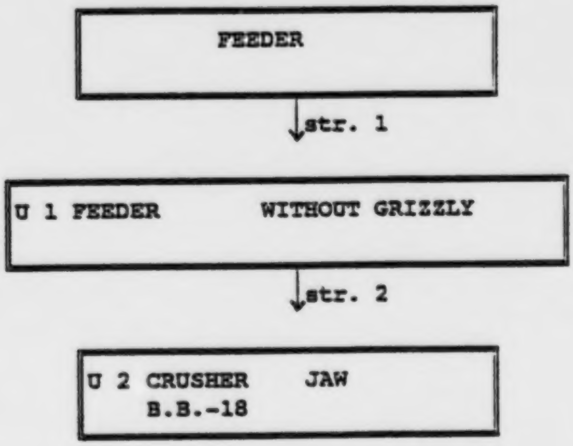
5. % of Humidity (H2O)      6.3 %


6. Tph (metric) to process :    48  tonnes / hour



       SELECTION: [ ]   <ret>continue  - (#)correction
```

FIGURE 4


FEEDER
str. 1
U 1 FEEDER WITHOUT GRIZZLY
str. 2
U 2 CRUSHER JAW
B.B.-18

```
D)escription - G)ranulomytrie - U)nit  - <ret>continue  [ ]
```

```
                   str.  1          str.  2
610   mm          100.0 %          100.0 %
305   mm           62.0 %           62.0 %
203   mm           40.0 %           40.0 %
152   mm           29.0 %           29.0 %
102   mm           18.0 %           18.0 %
 89   mm           15.0 %           15.0 %
 76   mm           12.0 %           12.0 %
 63   mm            9.0 %            9.0 %
 51   mm            7.0 %            7.0 %
 38   mm            5.0 %            5.0 %
 25   mm            4.0 %            4.0 %
 19   mm            3.0 %            3.0 %
 12.7 mm            2.0 %            2.0 %
  9.5 mm            1.5 %            1.5 %
  4 mesh            1.0 %            1.0 %
  8 mesh             .7 %             .7 %
 16 mesh             .4 %             .4 %
 30 mesh             .0 %             .0 %
 50 mesh             .0 %             .0 %
200 mesh             .0 %             .0 %
  Tph                45  tonnes       45  tonnes
  Water               3  tonnes        3  tonnes
Gr. max.            457 mm           457 mm
 D)escription - <ret> continue :
```

FIGURE 5

```
                         DESCRIPTION of unit no.      2

                                         - CRUSHER    JAW


           1. Model:   B.B.-18

           2. Capacity:    200  tonnes

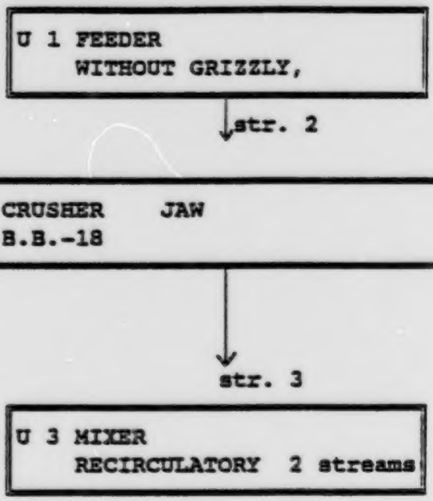
           3. Maximum opening:       457 mm

           4. Setting:    38 mm

              Efficiency :   38.0 %
              R/Reduction:   18.1
              Energy consumption:        24.2 kwh


   G)ranulomytrie - <ret>continue  [ ]
```

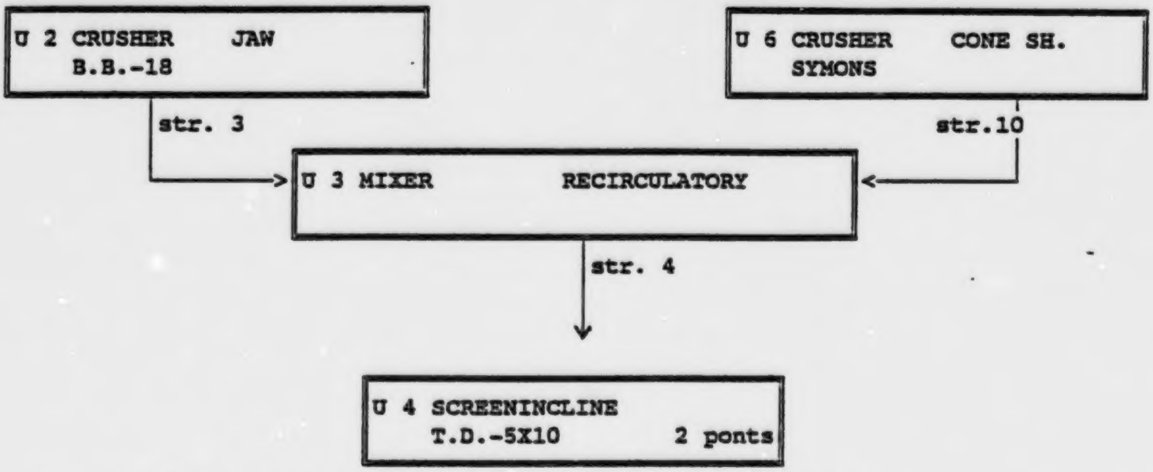
FIGURE 6


U 1 FEEDER
WITHOUT GRIZZLY,
str. 2
U 2 CRUSHER JAW
B.B.-18
str. 3
U 3 MIXER
RECIRCULATORY 2 streams

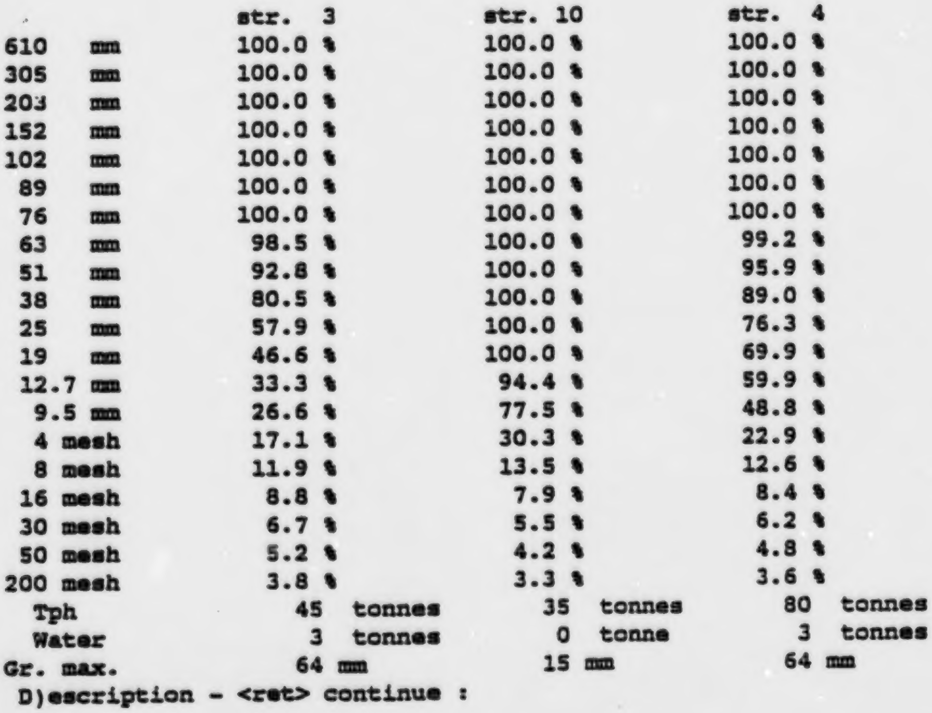
```
D)escription - G)ranulomytrie - U)nit  - <ret>continue  [ ]
```

```
                    str.  2          str.  3
610    mm          100.0 %          100.0 %
305    mm           62.0 %          100.0 %
203    mm           40.0 %          100.0 %
152    mm           29.0 %          100.0 %
102    mm           18.0 %          100.0 %
 89    mm           15.0 %          100.0 %
 76    mm           12.0 %          100.0 %
 63    mm            9.0 %           98.5 %
 51    mm            7.0 %           92.8 %
 38    mm            5.0 %           80.5 %
 25    mm            4.0 %           57.9 %
 19    mm            3.0 %           46.6 %
 12.7 mm             2.0 %           33.3 %
  9.5 mm             1.5 %           26.6 %
  4 mesh             1.0 %           17.1 %
  8 mesh              .7 %           11.9 %
 16 mesh              .4 %            8.8 %
 30 mesh              .0 %            6.7 %
 50 mesh              .0 %            5.2 %
200 mesh              .0 %            3.8 %
  Tph                 45  tonnes       45  tonnes
  Water                3  tonnes        3  tonnes
Gr. max.             457 mm            64 mm
 D)escription - <ret> continue :
```

FIGURE 7

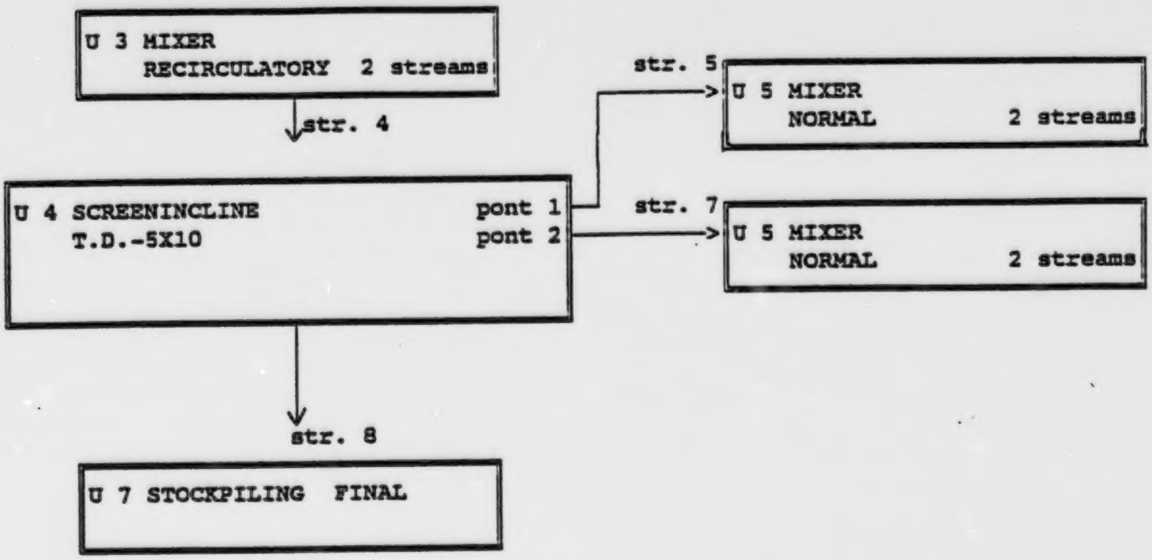

U 2 CRUSHER JAW
B.B.-18
U 6 CRUSHER CONE SH.
SYMONS
str. 3
str.10
U 3 MIXER RECIRCULATORY
str. 4
U 4 SCREENINCLINE
T.D.-5X10 2 ponts

```
D)escription - G)ranulomytrie - U)nit  - <ret>continue  [ ]
```



```
                str.  3          str. 10          str.  4
610    mm       100.0 %          100.0 %          100.0 %
305    mm       100.0 %          100.0 %          100.0 %
203    mm       100.0 %          100.0 %          100.0 %
152    mm       100.0 %          100.0 %          100.0 %
102    mm       100.0 %          100.0 %          100.0 %
 89    mm       100.0 %          100.0 %          100.0 %
 76    mm       100.0 %          100.0 %          100.0 %
 63    mm        98.5 %          100.0 %           99.2 %
 51    mm        92.8 %          100.0 %           95.9 %
 38    mm        80.5 %          100.0 %           89.0 %
 25    mm        57.9 %          100.0 %           76.3 %
 19    mm        46.6 %          100.0 %           69.9 %
  12.7 mm        33.3 %           94.4 %           59.9 %
   9.5 mm        26.6 %           77.5 %           48.8 %
   4 mesh        17.1 %           30.3 %           22.9 %
   8 mesh        11.9 %           13.5 %           12.6 %
  16 mesh         8.8 %            7.9 %            8.4 %
  30 mesh         6.7 %            5.5 %            6.2 %
  50 mesh         5.2 %            4.2 %            4.8 %
 200 mesh         3.8 %            3.3 %            3.6 %
  Tph              45  tonnes       35  tonnes       80  tonnes
  Water             3  tonnes        0  tonne         3  tonnes
Gr. max.           64 mm            15 mm            64 mm
 D)escription - <ret> continue :
```

FIGURE 8


U 3 MIXER
RECIRCULATORY 2 streams
str. 4
U 4 SCREENINCLINE
T.D.-5X10
pont 1
pont 2
str. 5
U 5 MIXER
NORMAL 2 streams
str. 7
U 5 MIXER
NORMAL 2 streams
str. 8
U 7 STOCKPILING FINAL

```
D)escription - G)ranulomytrie - U)nit  - <ret>continue  [ ]
```

```
           str.  4   str.  5   str.  7   str.  8
610    mm 100.0 %   100.0 %   100.0 %   100.0 %
305    mm 100.0 %   100.0 %   100.0 %   100.0 %
203    mm 100.0 %   100.0 %   100.0 %   100.0 %
152    mm 100.0 %   100.0 %   100.0 %   100.0 %
102    mm 100.0 %   100.0 %   100.0 %   100.0 %
 89    mm 100.0 %   100.0 %   100.0 %   100.0 %
 76    mm 100.0 %   100.0 %   100.0 %   100.0 %
 63    mm  99.2 %    97.0 %   100.0 %   100.0 %
 51    mm  95.9 %    84.5 %   100.0 %   100.0 %
 38    mm  89.0 %    58.3 %   100.0 %   100.0 %
 25    mm  76.3 %    14.0 %    94.1 %   100.0 %
 19    mm  69.9 %     3.4 %    73.3 %   100.0 %
 12.7 mm   59.9 %      .6 %    26.5 %    97.9 %
  9.5 mm   48.8 %      .2 %     4.5 %    85.2 %
  4 mesh   22.9 %      .1 %      .1 %    40.7 %
  8 mesh   12.6 %      .0 %      .0 %    22.4 %
 16 mesh    8.4 %      .0 %      .0 %    14.9 %
 30 mesh    6.2 %      .0 %      .0 %    11.0 %
 50 mesh    4.8 %      .0 %      .0 %     8.5 %
200 mesh    3.6 %      .0 %      .0 %     6.4 %
  Tph        80        21        14         45  tonnes
  Water       3         0         0          3  tonnes
Gr. max.     64 mm     64 mm     38 mm      19 mm
 D)escription - <ret> continue :
```

FIGURE 9

```
                         DESCRIPTION of unit no.      4

                                        - SCREEN/INCLINED, 2 DECKS

1. Model:  T.D.-5X10        Deck no.1 3. Water:      0 litre  per minute

2. Angle :  20                        4. Numb of sections:  1


                         SECTION 1

 Length:              5. 3048 mm

 Width:               6. 1524 mm

 Wire opening:        7.   25 mm

 Wire diameter:       8.   6.350 mm
  (0 = slotted)

 Deck efficiency:     99.6 %
<ret> to continue
```

```
                         DESCRIPTION of unit no.      4

                                        - SCREEN/INCLINED, 2 DECKS

1. Model:  T.D.-5X10        Deck no.2 3. Water:      0 litre  per minute

2. Angle :  20                        4. Numb of sections:  1


                         SECTION 1

 Length:              5. 3048 mm

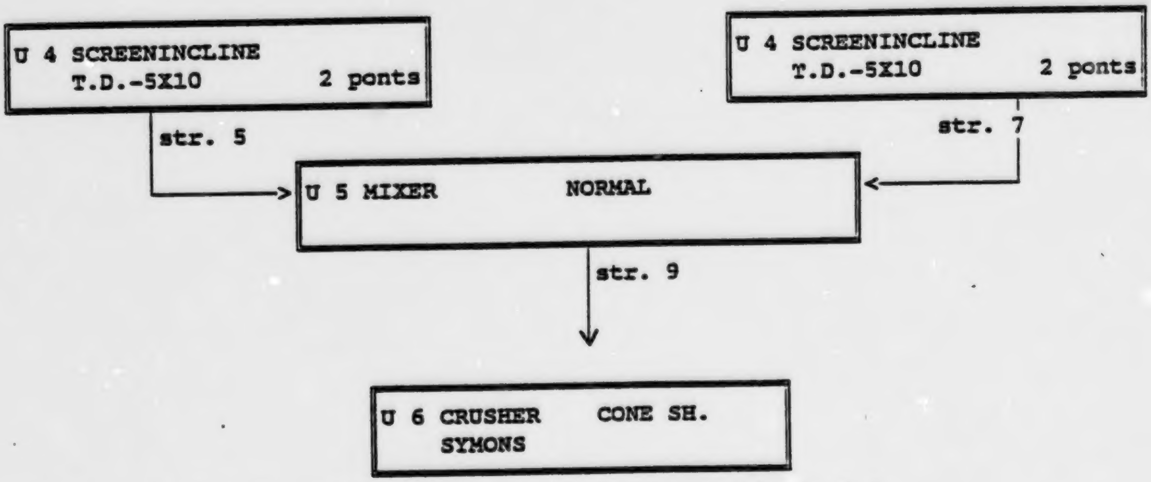
 Width:               6. 1524 mm

 Wire opening:        7.   13 mm

 Wire diameter:       8.    .000 mm
  (0 = slotted)

 Deck efficiency:     97.1 %
 G)ranulomytrie - <ret>continue  [ ]
```

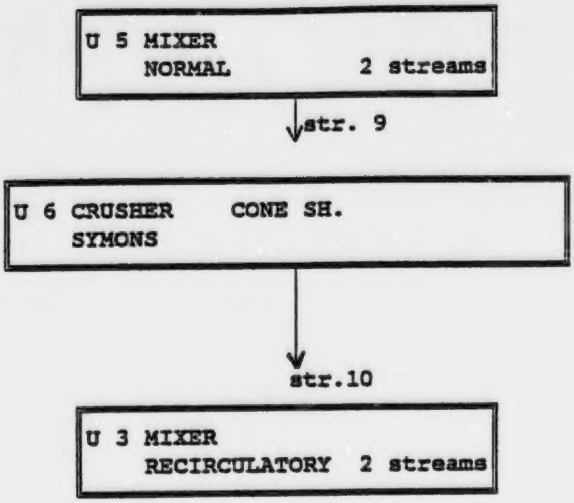
FIGURE 10


U 4 SCREENINCLINE
T.D.-5X10 2 ponts
U 4 SCREENINCLINE
T.D.-5X10 2 ponts
str. 5
str. 7
U 5 MIXER NORMAL
str. 9
U 6 CRUSHER CONE SH.
SYMONS

```
D)escription - G)ranulomytrie - U)nit  - <ret>continue  [ ]
```

```
                   str.  5        str.  7        str.  9
610    mm         100.0 %        100.0 %        100.0 %
305    mm         100.0 %        100.0 %        100.0 %
203    mm         100.0 %        100.0 %        100.0 %
152    mm         100.0 %        100.0 %        100.0 %
102    mm         100.0 %        100.0 %        100.0 %
 89    mm         100.0 %        100.0 %        100.0 %
 76    mm         100.0 %        100.0 %        100.0 %
 63    mm          97.0 %        100.0 %         98.2 %
 51    mm          84.5 %        100.0 %         90.6 %
 38    mm          58.3 %        100.0 %         74.8 %
 25    mm          14.0 %         94.1 %         45.8 %
 19    mm           3.4 %         73.3 %         31.1 %
 12.7 mm             .6 %         26.5 %         10.9 %
  9.5 mm             .2 %          4.5 %          1.9 %
  4 mesh             .1 %           .1 %           .1 %
  8 mesh             .0 %           .0 %           .0 %
 16 mesh             .0 %           .0 %           .0 %
 30 mesh             .0 %           .0 %           .0 %
 50 mesh             .0 %           .0 %           .0 %
200 mesh             .0 %           .0 %           .0 %
  Tph               21  tonnes    14  tonnes     35  tonnes
  Water              0  tonne      0  tonne       0  tonne
Gr. max.            64 mm         38 mm          64 mm
 D)escription - <ret> continue :
```

FIGURE 11


U 5 MIXER
NORMAL 2 streams
str. 9
U 6 CRUSHER CONE SH.
SYMONS
str.10
U 3 MIXER
RECIRCULATORY 2 streams

```
D)escription - G)ranulomytrie - U)nit  - <ret>continue  [ ]
```

```
                     str.  9          str. 10
610    mm           100.0 %          100.0 %
305    mm           100.0 %          100.0 %
203    mm           100.0 %          100.0 %
152    mm           100.0 %          100.0 %
102    mm           100.0 %          100.0 %
 89    mm           100.0 %          100.0 %
 76    mm           100.0 %          100.0 %
 63    mm            98.2 %          100.0 %
 51    mm            90.6 %          100.0 %
 38    mm            74.8 %          100.0 %
 25    mm            45.8 %          100.0 %
 19    mm            31.1 %          100.0 %
 12.7 mm             10.9 %           94.4 %
  9.5 mm              1.9 %           77.5 %
  4 mesh               .1 %           30.3 %
  8 mesh               .0 %           13.5 %
 16 mesh               .0 %            7.9 %
 30 mesh               .0 %            5.5 %
 50 mesh               .0 %            4.2 %
200 mesh               .0 %            3.3 %
  Tph                  35  tonnes       35  tonnes
  Water                 0  tonne         0  tonne
Gr. max.               64 mm            15 mm
 D)escription - <ret> continue :
```

FIGURE 12

```
                    DESCRIPTION of unit no.      6

                                  - CRUSHER    CONE SH.


        1. Model:   SYMONS

        2. Capacity:    100  tonnes

        3. Maximum opening:        64 mm

        4. Setting:    10 mm

        5. Cavity:  3  (1.extra fine, 2.fine, 3.medi
                          4."coarse", 5.extra "coarse")

           Efficiency :   58.7 %
           R/Reduction:    3.8
           Energy consumption:         17.3 kwh

G)ranulomytrie - <ret>continue  [ ]
```

FIGURE 13

PARTICLE SIZE ANALYSIS

SAMPLE NAME DOMINION EXPLORERS
ROD MILL FEED

DATA			CALCULATIONS	
PARTICLE SIZE RANGE (micrometer)	SAMPLE WEIGHT (grams)	METAL ASSAY (%)	WEIGHT DISTRIBUTION	WEIGHT PASSING
+19.0	31.5	0.00	4.1	95.9
12.7-19.0	122.1	0.00	15.9	80.0
9.5-12.7	137.6	0.00	17.9	62.2
4 mesh	183.8	0.00	23.9	38.3
8 mesh	104.2	0.00	13.5	24.7
16 mesh	59.4	0.00	7.7	17.0
30 mesh	32.2	0.00	4.2	12.8
50 mesh	20.6	0.00	2.7	10.1
-50	78.1	0.00	10.1	0.0
NA	0.0	0.00	0.0	
TOTALS	769.5		100	

APPENDIX 4

CRSHEX Simulator Verification

PROCESS STUDIES GROUP

technical notes

Project No.		*File No.*	
Project Plant Simulators		*Date* 05/08/88	*Page* 1

Subject: CRSHEX Simulator Package

Author(s): Donald S. Taylor

Objective: In the past few years, simulators have become available from CANMET to look at unit processes in metallurgical plants. The Department of Natural Resources and Energy has obtained these simulators for use in determining improvement in current plant operating conditions and for simulation of data for new mineral processing plants. In order to prove these simulators as a useful tool, it was determined that they be tested on processing units already operating in the area.

This is the second test in the program to verify the crushing simulator "CRSHEX". Brunswick Mining and Smelting in Bathurst has supplied us with the data needed to verify a simulation of their crushing plant.

Conclusions: The data from the Brunswick Mining crusher circuit was run on the simulator and the results of the final product from the simulator were different from the crushing plant operating data. A graph of the screen analysis of the final product from the plant and CRSHEX simulator are shown in Figure 1. The graph shows a large error in size distribution in the 2,380 micron to 19,700 micron range.

From the data of two tests run on in-province operating plants, it appears that the simulator does not work well at producing a reasonable screen analysis of the final crusher product. the program, however, can easily verify the correct selection of equipment, and approximate tonnage and recirculating loads in the crusher circuit.

Author(s) D. Taylor	***technical notes***	*Project No.*	*Page* 2	*of* 2

Discussion: The data on the product screen analysis from the CRSHEX simulator and the plant are considerably different. The reason for this error is most likely due to the unusual screen decks that are used in the plant. The normal screen is a conventional square design (1" x 1" squares) but in this plant, a screen of round punched holes of 1 1/8" diameter are used instead of square holes. The holes are placed in an irregular pattern on the screen deck with very large spaces between them.

The screen model in the CRSHEX program does not appear to take into account screens with round holes. The computer program can only take data for square screen holes. In consulting the "Handbook of Mineral Dressing" by A. F. Taggart (1960), a table was shown to convert 1 1/8" round holes to the equivalent of 7/8" square holes for screening parameters. The value of 7/8" was used in the final crushing simulator, but did not appear to change the final product sizing.

A second possible source of error was with the screen design. Most screens are designed so that the deck of the screen is at one angle from the top to bottom. This screen, however, has two angles; the first part of the deck at 15° and the second at 5°. The simulator could not be changed to take into account two screen angles. We were able to simulate this screen operating by breaking each screen into two separate screening operations; one screen at 15° and one at 5°. The simulator was tested with the new system of screens, but the results for the final product were similar to running the whole screen at 15° (see Figure 27 and Figures 28-30).

The data for the simulator for Brunswick Mining and Smelting is shown in Figures 2 to 27, and the ore flow diagram and final product screen analysis received from Brunswick Mining is shown in Figures 31 to 34.

The simulator package appears to work well, but is somewhat restricted when nonstandard operating parameters are used. The program appears to handle equipment and ore rate flow data reasonably well. This makes the program adequate for preliminary plant design engineering, and therefore preliminary plant costing.

FIGURE 1

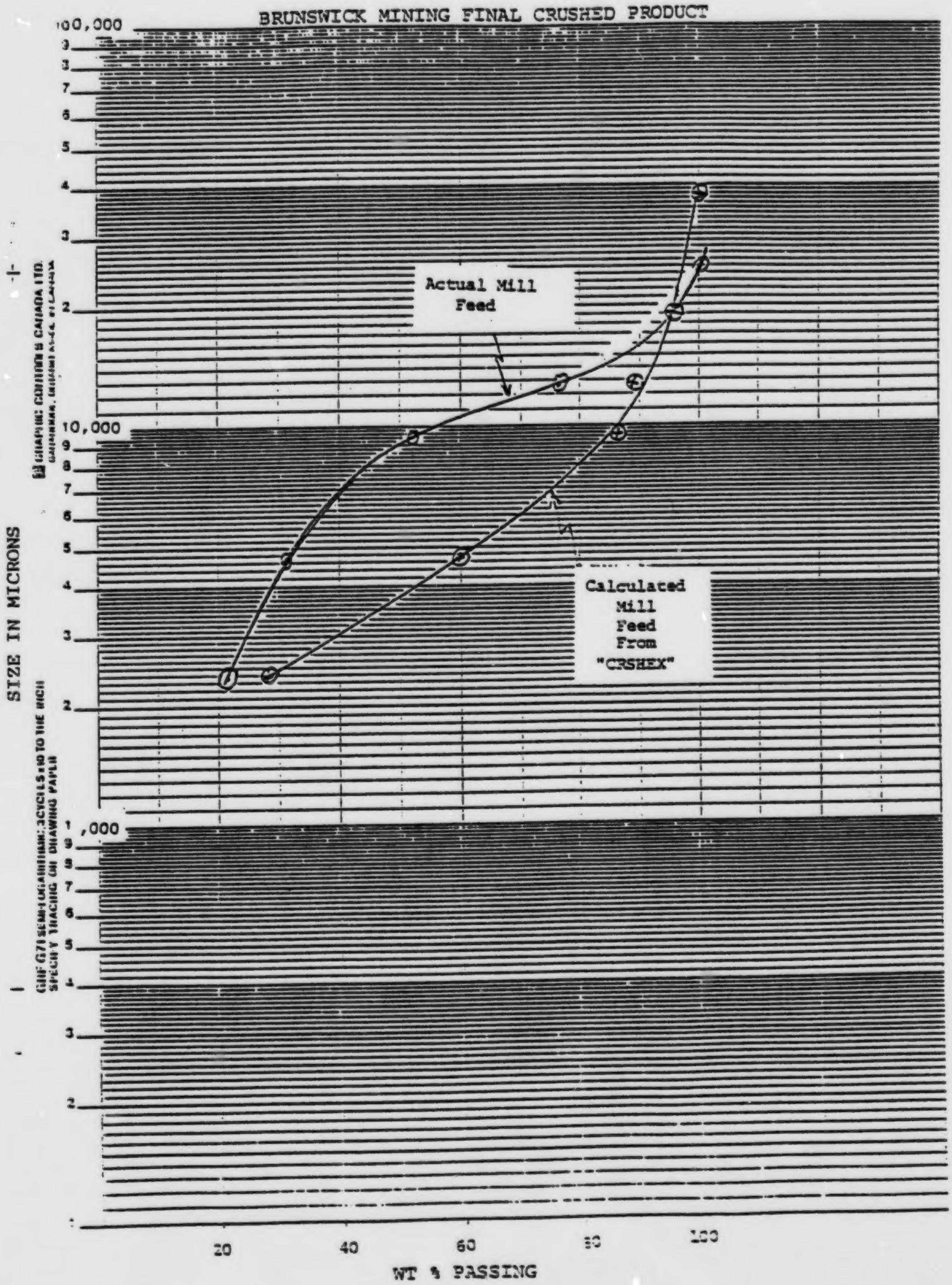

BRUNSWICK MINING FINAL CRUSHED PRODUCT
100,000
10,000
1,000
SIZE IN MICRONS
WT % PASSING
20
40
60
80
100
Actual Mill Feed
Calculated Mill Feed From "CRSHEX"
GRAPHIC CONTROLS CANADA LTD.

FIGURE 2

```
                                                              11/ 7/88

                        CRUSHER SIMULATION PACKAGE


                         USER IDENTIFICATION

                        1. Name:  BRUNSWICK MINING

                        2. Reference number:   2

                        3. Date:  11/ 7/88


SELECTION:   <ret>continue  - (#)correction - E)nd simulation
```

```
                              Feed data

1. Type of Material      :  Ore

2. Condition of Material:  WET

3. Density of Material   :    4.300    tonnes / cubic metre

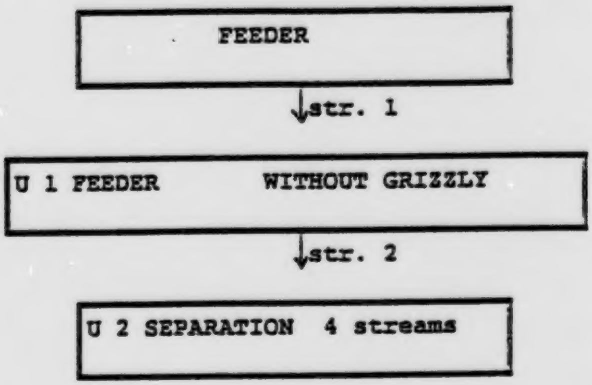
4. Bond Index (WI)   :       12.500  Kwh / tonne

5. % of Humidity (H2O)       4.0 %


6. Tph (metric) to process :   420  tonnes / hour


         SELECTION: [ ]   <ret>continue   - (#)correction
```

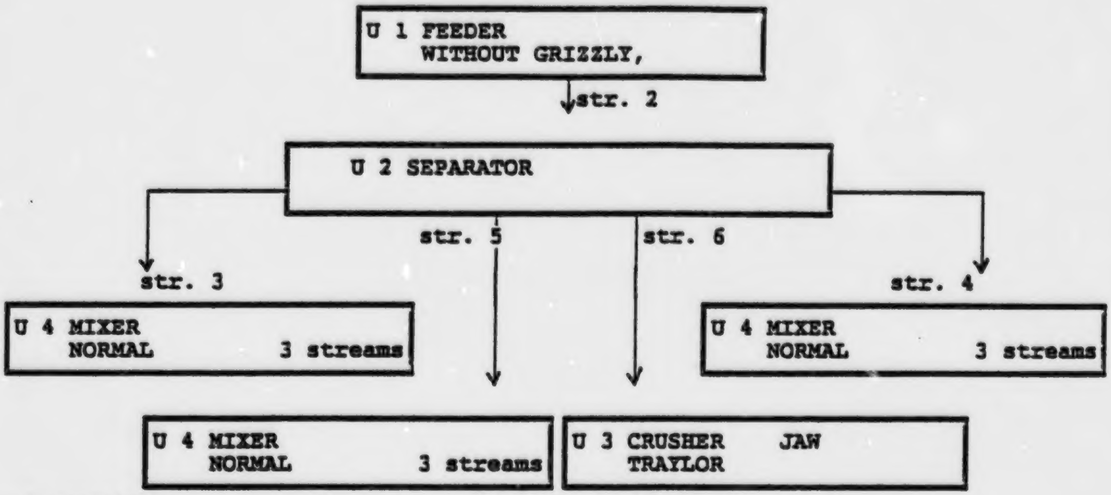
FIGURE 3


FEEDER
str. 1
U 1 FEEDER WITHOUT GRIZZLY
str. 2
U 2 SEPARATION 4 streams

```
D)escription - G)ranulomytrie - U)nit  - <ret>continue  [ ]
```

```
                    str.  1           str.  2
610    mm            61.0 %            61.0 %
305    mm            29.0 %            29.0 %
203    mm            18.0 %            18.0 %
152    mm            12.0 %            12.0 %
102    mm             7.0 %             7.0 %
 89    mm             6.0 %             6.0 %
 76    mm             5.0 %             5.0 %
 63    mm             4.5 %             4.5 %
 51    mm             4.0 %             4.0 %
 38    mm             3.0 %             3.0 %
 25    mm             2.0 %             2.0 %
 19    mm             1.5 %             1.5 %
  12.7 mm             1.0 %             1.0 %
   9.5 mm              .2 %              .2 %
   4 mesh              .0 %              .0 %
   8 mesh              .0 %              .0 %
  16 mesh              .0 %              .0 %
  30 mesh              .0 %              .0 %
  50 mesh              .0 %              .0 %
 200 mesh              .0 %              .0 %
  Tph                 403  tonnes       403  tonnes
  Water                17  tonnes        17  tonnes
 Gr. max.             914 mm            914 mm
 D)escription - <ret> continue :
```

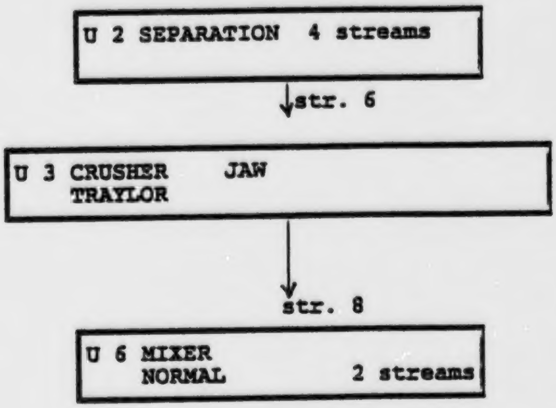
FIGURE 4


U 1 FEEDER
WITHOUT GRIZZLY,
str. 2
U 2 SEPARATOR
str. 5
str. 6
str. 3
str. 4
U 4 MIXER
NORMAL 3 streams
U 4 MIXER
NORMAL 3 streams
U 4 MIXER
NORMAL 3 streams
U 3 CRUSHER JAW
TRAYLOR

```
D)escription - G)ranulomytrie - U)nit  - <ret>continue  [ ]
```

```
               str.  2   str.  3   str.  4   str.  5   str.  6
610      mm   61.0 %    61.0 %    61.0 %    61.0 %    61.0 %
305      mm   29.0 %    29.0 %    29.0 %    29.0 %    29.0 %
203      mm   18.0 %    18.0 %    18.0 %    18.0 %    18.0 %
152      mm   12.0 %    12.0 %    12.0 %    12.0 %    12.0 %
102      mm    7.0 %     7.0 %     7.0 %     7.0 %     7.0 %
 89      mm    6.0 %     6.0 %     6.0 %     6.0 %     6.0 %
 76      mm    5.0 %     5.0 %     5.0 %     5.0 %     5.0 %
 63      mm    4.5 %     4.5 %     4.5 %     4.5 %     4.5 %
 51      mm    4.0 %     4.0 %     4.0 %     4.0 %     4.0 %
 38      mm    3.0 %     3.0 %     3.0 %     3.0 %     3.0 %
 25      mm    2.0 %     2.0 %     2.0 %     2.0 %     2.0 %
 19      mm    1.5 %     1.5 %     1.5 %     1.5 %     1.5 %
 12.7 mm       1.0 %     1.0 %     1.0 %     1.0 %     1.0 %
  9.5 mm        .2 %      .2 %      .2 %      .2 %      .2 %
  4 mesh        .0 %      .0 %      .0 %      .0 %      .0 %
  8 mesh        .0 %      .0 %      .0 %      .0 %      .0 %
 16 mesh        .0 %      .0 %      .0 %      .0 %      .0 %
 30 mesh        .0 %      .0 %      .0 %      .0 %      .0 %
 50 mesh        .0 %      .0 %      .0 %      .0 %      .0 %
200 mesh        .0 %      .0 %      .0 %      .0 %      .0 %
  Tph         403       101       101       101       101   tonnes
  Water        17         4         4         4         4   tonnes
Gr. max.      914 mm    914 mm    914 mm    914 mm    914 mm
 D)escription - <ret> continue :
```

FIGURE 5


U 2 SEPARATION 4 streams
str. 6
U 3 CRUSHER JAW
TRAYLOR
str. 8
U 6 MIXER
NORMAL 2 streams

```
D)escription - G)ranulomytrie - U)nit  - <ret>continue  [ ]
```

```
                     str.  6         str.  8
610    mm            61.0 %          100.0 %
305    mm            29.0 %          100.0 %
203    mm            18.0 %           99.8 %
152    mm            12.0 %           96.5 %
102    mm             7.0 %           78.6 %
 89    mm             6.0 %           70.6 %
 76    mm             5.0 %           61.9 %
 63    mm             4.5 %           53.1 %
 51    mm             4.0 %           44.2 %
 38    mm             3.0 %           35.4 %
 25    mm             2.0 %           26.6 %
 19    mm             1.5 %           22.1 %
 12.7 mm              1.0 %           15.7 %
  9.5 mm               .2 %           12.5 %
  4 mesh               .0 %            8.2 %
  8 mesh               .0 %            5.6 %
 16 mesh               .0 %            3.7 %
 30 mesh               .0 %            2.3 %
 50 mesh               .0 %            1.1 %
200 mesh               .0 %             .0 %
  Tph                 101   tonnes     101   tonnes
  Water                 4   tonnes       4   tonnes
Gr. max.              914 mm           221 mm
 D)escription - <ret> continue :
```

FIGURE 6

```
                    DESCRIPTION of unit no.      3
                                - CRUSHER    JAW

        1. Model:   TRAYLOR

        2. Capacity:    550  tonnes

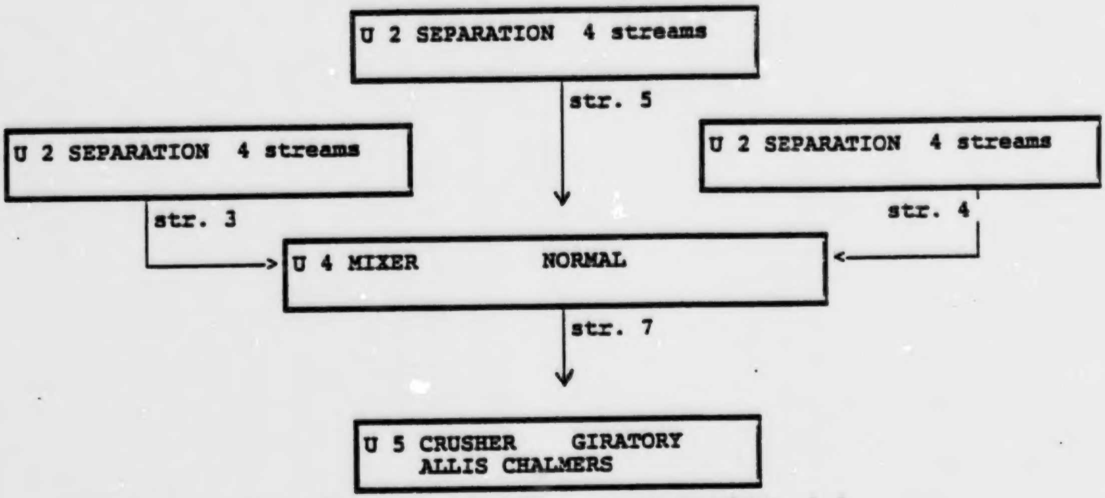
        3. Maximum opening:       914 mm

        4. Setting:   114 mm

           Efficiency :   49.2 %
           R/Reduction:    6.2
           Energy consumption:        19.6 kwh


G)ranulomytrie - <ret>continue  [ ]
```

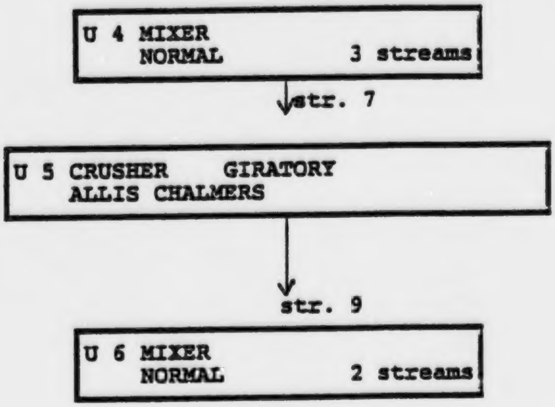
FIGURE 7


U 2 SEPARATION 4 streams
str. 5
U 2 SEPARATION 4 streams
U 2 SEPARATION 4 streams
str. 3
str. 4
U 4 MIXER NORMAL
str. 7
U 5 CRUSHER GIRATORY
ALLIS CHALMERS

```
D)escription - G)ranulomytrie - U)nit  - <ret>continue  [ ]
```

```
                 str.  3   str.  4   str.  5   str.  7
610    mm   61.0 %    61.0 %    61.0 %    61.0 %
305    mm   29.0 %    29.0 %    29.0 %    29.0 %
203    mm   18.0 %    18.0 %    18.0 %    18.0 %
152    mm   12.0 %    12.0 %    12.0 %    12.0 %
102    mm    7.0 %     7.0 %     7.0 %     7.0 %
89     mm    6.0 %     6.0 %     6.0 %     6.0 %
76     mm    5.0 %     5.0 %     5.0 %     5.0 %
63     mm    4.5 %     4.5 %     4.5 %     4.5 %
51     mm    4.0 %     4.0 %     4.0 %     4.0 %
38     mm    3.0 %     3.0 %     3.0 %     3.0 %
25     mm    2.0 %     2.0 %     2.0 %     2.0 %
19     mm    1.5 %     1.5 %     1.5 %     1.5 %
12.7 mm      1.0 %     1.0 %     1.0 %     1.0 %
  9.5 mm      .2 %      .2 %      .2 %      .2 %
   4 mesh     .0 %      .0 %      .0 %      .0 %
   8 mesh     .0 %      .0 %      .0 %      .0 %
  16 mesh     .0 %      .0 %      .0 %      .0 %
  30 mesh     .0 %      .0 %      .0 %      .0 %
  50 mesh     .0 %      .0 %      .0 %      .0 %
 200 mesh     .0 %      .0 %      .0 %      .0 %
  Tph       101       101       101       302  tonnes
  Water       4         4         4        13  tonnes
 Gr. max.   914 mm    914 mm    914 mm    914 mm
  D)escription - <ret> continue :
```

FIGURE 8


U 4 MIXER
NORMAL 3 streams
str. 7
U 5 CRUSHER GIRATORY
ALLIS CHALMERS
str. 9
U 6 MIXER
NORMAL 2 streams

```
D)escription - G)ranulomytrie - U)nit  - <ret>continue  [ ]
```

```
                      str.  7          str.  9
610    mm             61.0 %           100.0 %
305    mm             29.0 %           100.0 %
203    mm             18.0 %           100.0 %
152    mm             12.0 %           100.0 %
102    mm              7.0 %            97.1 %
 89    mm              6.0 %            94.3 %
 76    mm              5.0 %            87.7 %
 63    mm              4.5 %            77.6 %
 51    mm              4.0 %            65.1 %
 38    mm              3.0 %            51.5 %
 25    mm              2.0 %            37.9 %
 19    mm              1.5 %            31.1 %
 12.7 mm               1.0 %            22.4 %
  9.5 mm                .2 %            18.0 %
   4 mesh               .0 %            11.9 %
   8 mesh               .0 %             8.3 %
  16 mesh               .0 %             6.0 %
  30 mesh               .0 %             4.4 %
  50 mesh               .0 %             3.1 %
 200 mesh               .0 %             1.8 %
  Tph                 302  tonnes      302  tonnes
  Water                13  tonnes       13  tonnes
Gr. max.              914 mm           134 mm
 D)escription - <ret> continue :
```

FIGURE 9

```
                         DESCRIPTION of unit no.      5
                                   - CRUSHER    GIRATORY

          1. Model:   ALLIS C

          2. Capacity:   2000  tonnes

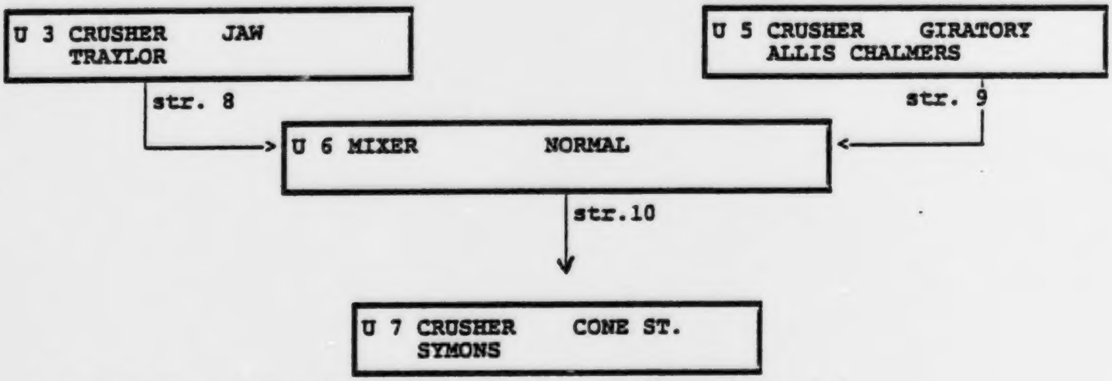
          3. Maximum opening:        914 mm

          4. Setting:    76 mm

             Efficiency :   40.6 %
             R/Reduction:   12.6
             Energy consumption:        100.4 kwh


     G)ranulomytrie - <ret>continue  [ ]
```

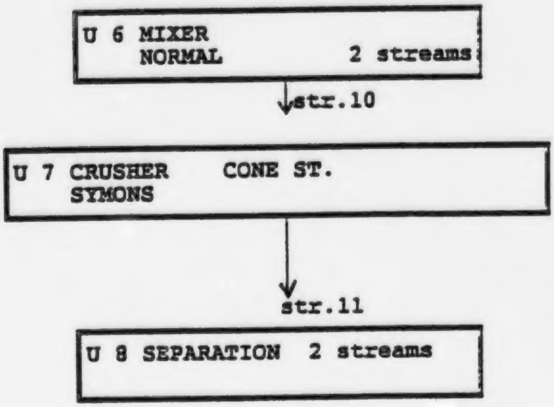
FIGURE 10


U 3 CRUSHER JAW
TRAYLOR
U 5 CRUSHER GIRATORY
ALLIS CHALMERS
str. 8
str. 9
U 6 MIXER NORMAL
str.10
U 7 CRUSHER CONE ST.
SYMONS

```
D)escription - G)ranulomytrie - U)nit  - <ret>continue  [ ]
```

```
                    str.  8          str.  9          str. 10
   610    mm       100.0 %          100.0 %          100.0 %
   305    mm       100.0 %          100.0 %          100.0 %
   203    mm        99.8 %          100.0 %           99.9 %
   152    mm        96.5 %          100.0 %           99.1 %
   102    mm        78.6 %           97.1 %           92.5 %
    89    mm        70.6 %           94.3 %           88.4 %
    76    mm        61.9 %           87.7 %           81.2 %
    63    mm        53.1 %           77.6 %           71.5 %
    51    mm        44.2 %           65.1 %           59.9 %
    38    mm        35.4 %           51.5 %           47.5 %
    25    mm        26.6 %           37.9 %           35.1 %
    19    mm        22.1 %           31.1 %           28.8 %
    12.7 mm         15.7 %           22.4 %           20.7 %
     9.5 mm         12.5 %           18.0 %           16.6 %
     4 mesh          8.2 %           11.9 %           11.0 %
     8 mesh          5.6 %            8.3 %            7.6 %
    16 mesh          3.7 %            6.0 %            5.4 %
    30 mesh          2.3 %            4.4 %            3.9 %
    50 mesh          1.1 %            3.1 %            2.6 %
   200 mesh           .0 %            1.8 %            1.3 %
     Tph            101   tonnes     302   tonnes     403   tonne
     Water            4   tonnes      13   tonnes      17   tonne
   Gr. max.         221 mm           134 mm           221 mm
    D)escription - <ret> continue :
```

FIGURE 11


U 6 MIXER
NORMAL
2 streams
str.10
U 7 CRUSHER
CONE ST.
SYMONS
str.11
U 8 SEPARATION
2 streams

```
D)escription - G)ranulomytrie - U)nit  - <ret>continue  [ ]


                    str. 10             str. 11
 610     mm         100.0 %             100.0 %
 305     mm         100.0 %             100.0 %
 203     mm          99.9 %             100.0 %
 152     mm          99.1 %             100.0 %
 102     mm          92.5 %             100.0 %
  89     mm          88.4 %             100.0 %
  76     mm          81.2 %             100.0 %
  63     mm          71.5 %              97.1 %
  51     mm          59.9 %              87.3 %
  38     mm          47.5 %              67.3 %
  25     mm          35.1 %              43.0 %
  19     mm          28.8 %              31.9 %
  12.7 mm            20.7 %              24.4 %
   9.5 mm            16.6 %              21.4 %
   4 mesh            11.0 %              17.4 %
   8 mesh             7.6 %              12.1 %
  16 mesh             5.4 %               9.0 %
  30 mesh             3.9 %               7.0 %
  50 mesh             2.6 %               5.5 %
 200 mesh             1.3 %               4.1 %
   Tph                403  tonnes         403  tonnes
   Water               17  tonnes          17  tonnes
 Gr. max.             221 mm               73 mm
  D)escription - <ret> continue :
```

FIGURE 12

```
                    DESCRIPTION of unit no.      7
                                   - CRUSHER    CONE ST.

      1. Model:   SYMONS

      2. Capacity:    500  tonnes

      3. Maximum opening:        221 mm

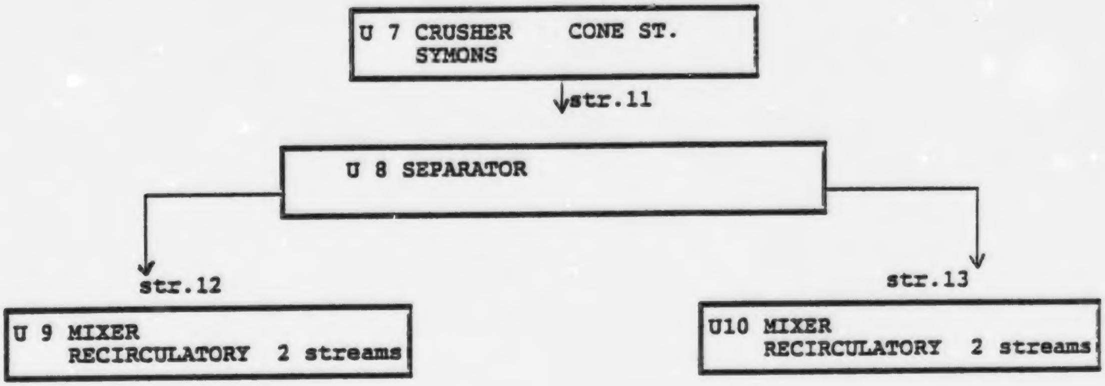
      4. Setting:    38 mm

      5. Cavity:  4  (1.extra fine, 2.fine, 3.medi
                       4."coarse", 5.extra "coarse")

         Efficiency :  100.0 %
         R/Reduction:    1.5
         Energy consumption:         46.2 kwh

G)ranulomytrie - <ret>continue  [ ]
```

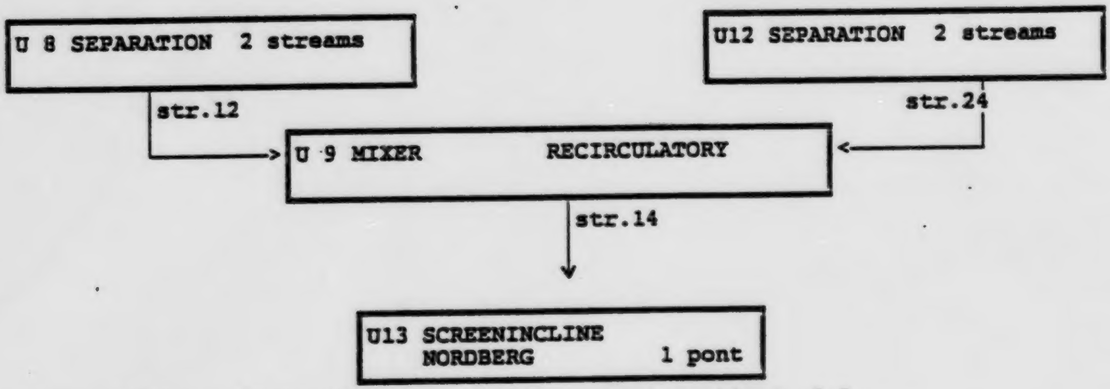
FIGURE 13


U 7 CRUSHER CONE ST.
SYMONS
str.11
U 8 SEPARATOR
str.12
str.13
U 9 MIXER
RECIRCULATORY 2 streams
U10 MIXER
RECIRCULATORY 2 streams

```
D)escription - G)ranulomytrie - U)nit  - <ret>continue  [ ]
```

```
                   str. 11          str. 12          str. 13
610    mm          100.0 %          100.0 %          100.0 %
305    mm          100.0 %          100.0 %          100.0 %
203    mm          100.0 %          100.0 %          100.0 %
152    mm          100.0 %          100.0 %          100.0 %
102    mm          100.0 %          100.0 %          100.0 %
 89    mm          100.0 %          100.0 %          100.0 %
 76    mm          100.0 %          100.0 %          100.0 %
 63    mm           97.1 %           97.1 %           97.1 %
 51    mm           87.3 %           87.3 %           87.3 %
 38    mm           67.3 %           67.3 %           67.3 %
 25    mm           43.0 %           43.0 %           43.0 %
 19    mm           31.9 %           31.9 %           31.9 %
 12.7  mm           24.4 %           24.4 %           24.4 %
  9.5  mm           21.4 %           21.4 %           21.4 %
  4 mesh            17.4 %           17.4 %           17.4 %
  8 mesh            12.1 %           12.1 %           12.1 %
 16 mesh             9.0 %            9.0 %            9.0 %
 30 mesh             7.0 %            7.0 %            7.0 %
 50 mesh             5.5 %            5.5 %            5.5 %
200 mesh             4.1 %            4.1 %            4.1 %
  Tph                403   tonnes     202   tonnes     202   tonnes
  Water               17   tonnes       9   tonnes       9   tonnes
Gr. max.              73 mm            73 mm            73 mm
 D)escription - <ret> continue :
```

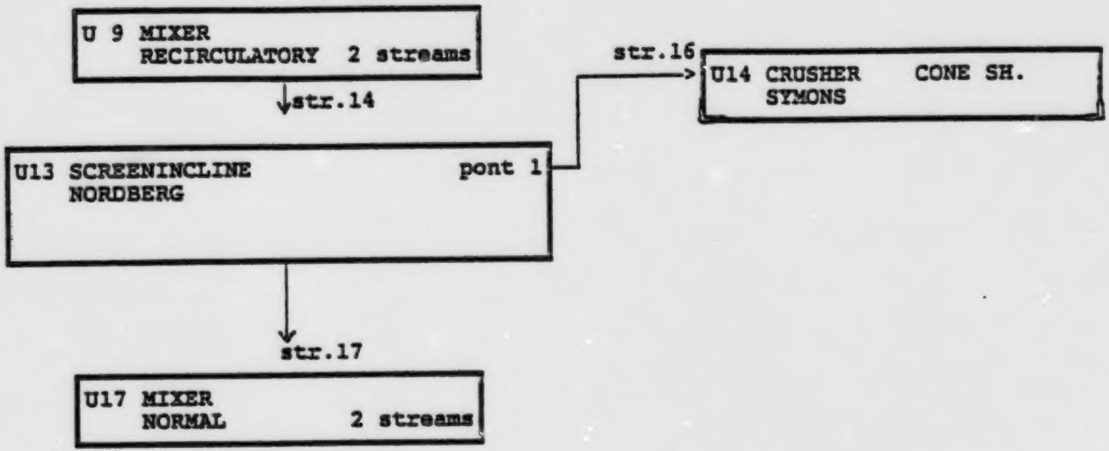
FIGURE 14


U 8 SEPARATION 2 streams
U12 SEPARATION 2 streams
str.12
str.24
U 9 MIXER RECIRCULATORY
str.14
U13 SCREENINCLINE
NORDBERG 1 pont

```
D)escription - G)ranulomytrie - U)nit  - <ret>continue  [ ]
```

```
                      str. 12        str. 24        str. 14
610    mm             100.0 %        100.0 %        100.0 %
305    mm             100.0 %        100.0 %        100.0 %
203    mm             100.0 %        100.0 %        100.0 %
152    mm             100.0 %        100.0 %        100.0 %
102    mm             100.0 %        100.0 %        100.0 %
 89    mm             100.0 %        100.0 %        100.0 %
 76    mm             100.0 %        100.0 %        100.0 %
 63    mm              97.1 %        100.0 %         98.2 %
 51    mm              87.3 %        100.0 %         92.3 %
 38    mm              67.3 %        100.0 %         80.1 %
 25    mm              43.0 %        100.0 %         65.3 %
 19    mm              31.9 %        100.0 %         58.6 %
 12.7 mm              24.4 %        100.0 %         54.0 %
   9.5 mm             21.4 %        100.0 %         52.2 %
    4 mesh            17.4 %         65.6 %         36.3 %
    8 mesh            12.1 %         24.7 %         17.0 %
   16 mesh             9.0 %         12.0 %         10.2 %
   30 mesh             7.0 %          7.3 %          7.1 %
   50 mesh             5.5 %          5.1 %          5.3 %
  200 mesh             4.1 %          3.5 %          3.9 %
  Tph                 202   tonnes   130   tonnes   331   tonnes
  Water                 9   tonnes     0   tonne      9   tonnes
Gr. max.              73 mm            9 mm          73 mm
 D)escription - <ret> continue :
```

FIGURE 15


U 9 MIXER
RECIRCULATORY 2 streams
str.14
U13 SCREENINCLINE
NORDBERG
pont 1
str.16
U14 CRUSHER CONE SH.
SYMONS
str.17
U17 MIXER
NORMAL 2 streams

```
D)escription - G)ranulomytrie - U)nit  - <ret>continue  [ ]
```

```
                     str. 14          str. 16          str. 17
   610   mm          100.0 %          100.0 %          100.0 %
   305   mm          100.0 %          100.0 %          100.0 %
   203   mm          100.0 %          100.0 %          100.0 %
   152   mm          100.0 %          100.0 %          100.0 %
   102   mm          100.0 %          100.0 %          100.0 %
    89   mm          100.0 %          100.0 %          100.0 %
    76   mm          100.0 %          100.0 %          100.0 %
    63   mm           98.2 %           95.4 %          100.0 %
    51   mm           92.3 %           80.3 %          100.0 %
    38   mm           80.1 %           49.2 %          100.0 %
    25   mm           65.3 %           12.5 %           99.3 %
    19   mm           58.6 %            1.9 %           95.1 %
    12.7 mm           54.0 %             .1 %           88.7 %
     9.5 mm           52.2 %             .0 %           85.8 %
     4 mesh           36.3 %             .0 %           59.7 %
     8 mesh           17.0 %             .0 %           28.0 %
    16 mesh           10.2 %             .0 %           16.8 %
    30 mesh            7.1 %             .0 %           11.7 %
    50 mesh            5.3 %             .0 %            8.7 %
   200 mesh            3.9 %             .0 %            6.4 %
    Tph                331   tonnes    130   tonnes     202   tonnes
     Water               9   tonnes      0   tonne        9   tonnes
  Gr. max.              73 mm           73 mm            38 mm
   D)escription - <ret> continue :
```

FIGURE 16

```
                         DESCRIPTION of unit no.     13

                                       - SCREEN/INCLINED, 1 DECK

1. Model:  NORDBERG          Deck no.1 3. Water:      0 litre  per minute

2. Angle :  15                         4. Numb of sections:  1

                         SECTION 1

 Length:               5. 3048 mm

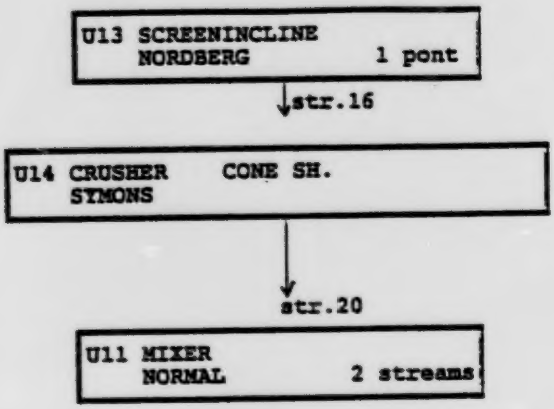
 Width:                6. 1524 mm

 Wire opening:         7.   22 mm

 Wire diameter:        8.  10.000 mm
  (0 = slotted)

 Deck efficiency:    100.0 %
 G)ranulomytrie - <ret>continue  [ ]
```

FIGURE 17


U13 SCREENINCLINE
NORDBERG 1 pont
str.16
U14 CRUSHER CONE SH.
SYMONS
str.20
U11 MIXER
NORMAL 2 streams

```
D)escription - G)ranulomytrie - U)nit  - <ret>continue  [ ]
```

```
                      str. 16          str. 20
610    mm             100.0 %          100.0 %
305    mm             100.0 %          100.0 %
203    mm             100.0 %          100.0 %
152    mm             100.0 %          100.0 %
102    mm             100.0 %          100.0 %
 89    mm             100.0 %          100.0 %
 76    mm             100.0 %          100.0 %
 63    mm              95.4 %          100.0 %
 51    mm              80.3 %          100.0 %
 38    mm              49.2 %          100.0 %
 25    mm              12.5 %          100.0 %
 19    mm               1.9 %          100.0 %
 12.7  mm                .1 %          100.0 %
  9.5  mm                .0 %          100.0 %
  4 mesh                 .0 %           65.6 %
  8 mesh                 .0 %           24.7 %
 16 mesh                 .0 %           12.0 %
 30 mesh                 .0 %            7.3 %
 50 mesh                 .0 %            5.1 %
200 mesh                 .0 %            3.5 %
  Tph                   130  tonnes
```

FIGURE 18

```
                    DESCRIPTION of unit no.     14
                                   - CRUSHER    CONE SH.

        1. Model:   SYMONS

        2. Capacity:    300  tonnes

        3. Maximum opening:        73 mm

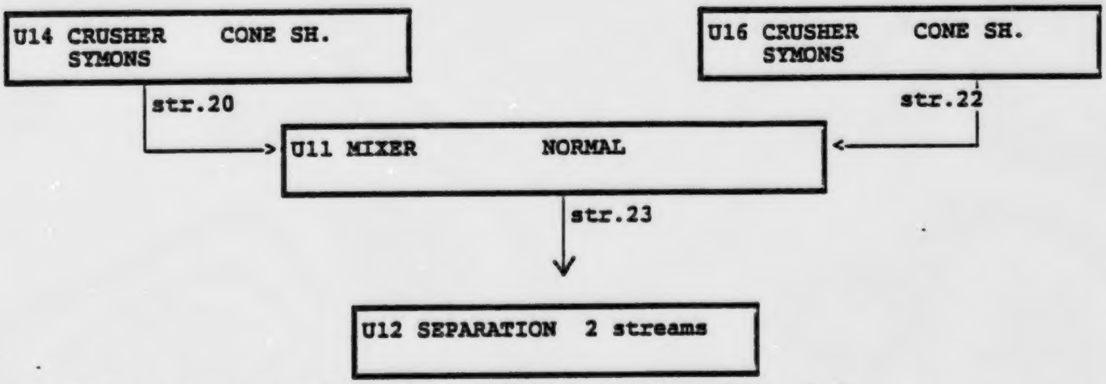
        4. Setting:     6 mm

        5. Cavity:  2  (1.extra fine, 2.fine, 3.medi
                         4."coarse", 5.extra "coarse")

           Efficiency :  100.0 %
           R/Reduction:    5.1
           Energy consumption:        104.1 kwh

  G)ranulomytrie - <ret>continue  [
```

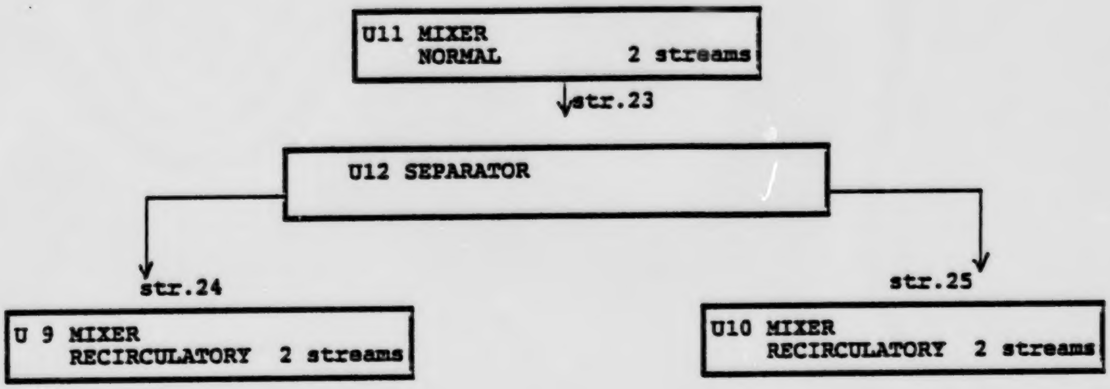
FIGURE 19


U14 CRUSHER CONE SH. SYMONS
U16 CRUSHER CONE SH. SYMONS
str.20
str.22
U11 MIXER NORMAL
str.23
U12 SEPARATION 2 streams

```
D)escription - G)ranulomytrie - U)nit  - <ret>continue  [ ]
```

```
                  str. 20           str. 22           str. 23
610    mm         100.0 %           100.0 %           100.0 %
305    mm         100.0 %           100.0 %           100.0 %
203    mm         100.0 %           100.0 %           100.0 %
152    mm         100.0 %           100.0 %           100.0 %
102    mm         100.0 %           100.0 %           100.0 %
 89    mm         100.0 %           100.0 %           100.0 %
 76    mm         100.0 %           100.0 %           100.0 %
 63    mm         100.0 %           100.0 %           100.0 %
 51    mm         100.0 %           100.0 %           100.0 %
 38    mm         100.0 %           100.0 %           100.0 %
 25    mm         100.0 %           100.0 %           100.0 %
 19    mm         100.0 %           100.0 %           100.0 %
 12.7  mm         100.0 %           100.0 %           100.0 %
  9.5  mm         100.0 %           100.0 %           100.0 %
  4 mesh           65.6 %            65.6 %            65.6 %
  8 mesh           24.7 %            24.7 %            24.7 %
 16 mesh           12.0 %            12.0 %            12.0 %
 30 mesh            7.3 %             7.3 %             7.3 %
 50 mesh            5.1 %             5.1 %             5.1 %
200 mesh            3.5 %             3.5 %             3.5 %
  Tph               130  tonnes       130  tonnes       260  tonnes
  Water               0  tonne          0  tonne          0  tonne
Gr. max.
```

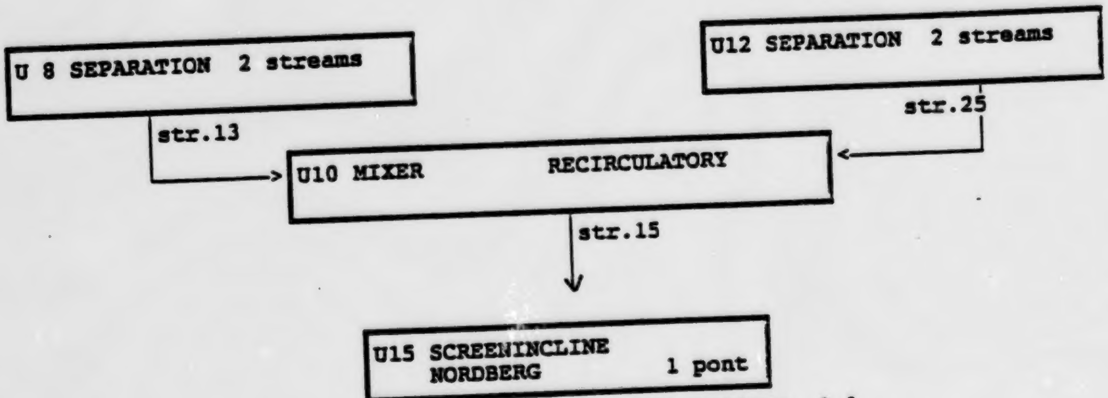
FIGURE 20


U11 MIXER
NORMAL 2 streams
str.23
U12 SEPARATOR
str.24
str.25
U 9 MIXER
RECIRCULATORY 2 streams
U10 MIXER
RECIRCULATORY 2 streams

```
D)escription - G)ranulomytrie - U)nit  - <ret>continue  [ ]
```

```
                 str. 23        str. 24        str. 25
610    mm        100.0 %        100.0 %        100.0 %
305    mm        100.0 %        100.0 %        100.0 %
203    mm        100.0 %        100.0 %        100.0 %
152    mm        100.0 %        100.0 %        100.0 %
102    mm        100.0 %        100.0 %        100.0 %
 89    mm        100.0 %        100.0 %        100.0 %
 76    mm        100.0 %        100.0 %        100.0 %
 63    mm        100.0 %        100.0 %        100.0 %
 51    mm        100.0 %        100.0 %        100.0 %
 38    mm        100.0 %        100.0 %        100.0 %
 25    mm        100.0 %        100.0 %        100.0 %
 19    mm        100.0 %        100.0 %        100.0 %
 12.7  mm        100.0 %        100.0 %        100.0 %
  9.5  mm        100.0 %        100.0 %        100.0 %
  4 mesh          65.6 %         65.6 %         65.6 %
  8 mesh          24.7 %         24.7 %         24.7 %
 16 mesh          12.0 %         12.0 %         12.0 %
 30 mesh           7.3 %          7.3 %          7.3 %
 50 mesh           5.1 %          5.1 %          5.1 %
200 mesh           3.5 %          3.5 %          3.5 %
  Tph            260  tonnes    130  tonnes    130  tonnes
  Water            0  tonne       0  tonne       0  tonne
Gr. max.           9  mm          9  mm          9  mm
 D)escription - <ret> continue :
```

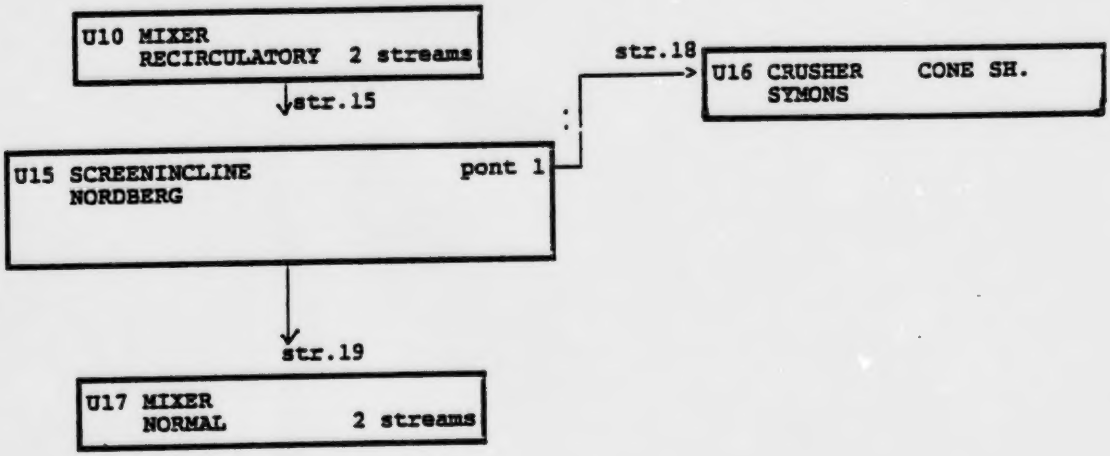
FIGURE 21


U 8 SEPARATION 2 streams
U12 SEPARATION 2 streams
str.13
str.25
U10 MIXER RECIRCULATORY
str.15
U15 SCREENINCLINE
NORDBERG 1 pont

```
D)escription - G)ranulomytrie - U)nit  - <ret>continue  [ ]
```

```
                  str. 13        str. 25        str. 15
610    mm         100.0 %        100.0 %        100.0 %
305    mm         100.0 %        100.0 %        100.0 %
203    mm         100.0 %        100.0 %        100.0 %
152    mm         100.0 %        100.0 %        100.0 %
102    mm         100.0 %        100.0 %        100.0 %
 89    mm         100.0 %        100.0 %        100.0 %
 76    mm         100.0 %        100.0 %        100.0 %
 63    mm          97.1 %        100.0 %         98.2 %
 51    mm          87.3 %        100.0 %         92.3 %
 38    mm          67.3 %        100.0 %         80.1 %
 25    mm          43.0 %        100.0 %         65.3 %
 19    mm          31.9 %        100.0 %         58.6 %
 12.7 mm           24.4 %        100.0 %         54.0 %
  9.5 mm           21.4 %        100.0 %         52.2 %
   4 mesh          17.4 %         65.6 %         36.3 %
   8 mesh          12.1 %         24.7 %         17.0 %
  16 mesh           9.0 %         12.0 %         10.2 %
  30 mesh           7.0 %          7.3 %          7.1 %
  50 mesh           5.5 %          5.1 %          5.3 %
 200 mesh           4.1 %          3.5 %          3.9 %
   Tph            202  tonnes   130  tonnes   331  tonnes
   Water            9  tonnes     0  tonne      9  tonnes
 Gr. max.          73 mm          9 mm         73 mm
  D)escription - <ret> continue :
```

FIGURE 22



U10 MIXER
RECIRCULATORY 2 streams
str.15
U15 SCREENINCLINE
NORDBERG
pont 1
str.18
U16 CRUSHER CONE SH.
SYMONS
str.19
U17 MIXER
NORMAL 2 streams

```
D)escription - G)ranulomytrie - U)nit  - <ret>continue  [ ]
```

```
                 str. 15          str. 18          str. 19
610    mm        100.0 %          100.0 %          100.0 %
305    mm        100.0 %          100.0 %          100.0 %
203    mm        100.0 %          100.0 %          100.0 %
152    mm        100.0 %          100.0 %          100.0 %
102    mm        100.0 %          100.0 %          100.0 %
 89    mm        100.0 %          100.0 %          100.0 %
 76    mm        100.0 %          100.0 %          100.0 %
 63    mm         98.2 %           95.4 %          100.0 %
 51    mm         92.3 %           80.3 %          100.0 %
 38    mm         80.1 %           49.2 %          100.0 %
 25    mm         65.3 %           12.5 %           99.3 %
 19    mm         58.6 %            1.9 %           95.1 %
 12.7  mm         54.0 %             .1 %           88.7 %
  9.5  mm         52.2 %             .0 %           85.8 %
   4 mesh         36.3 %             .0 %           59.7 %
   8 mesh         17.0 %             .0 %           28.0 %
  16 mesh         10.2 %             .0 %           16.8 %
  30 mesh          7.1 %             .0 %           11.7 %
  50 mesh          5.3 %             .0 %            8.7 %
 200 mesh          3.9 %             .0 %            6.4 %
   Tph             331   tonnes    130   tonnes     202   tonnes
   Water             9   tonnes      0   tonne        9   tonnes
Gr. max.            73 mm           73 mm            38 mm
 D)escription - <ret> continue :
```

FIGURE 23

```
                        DESCRIPTION of unit no.     15
                                     - SCREEN/INCLINED, 1 DECK

1. Model:  NORDBERG           Deck no.1 3. Water:      0 litre  per minute

2. Angle :  15                          4. Numb of sections:  1

                        SECTION 1

 Length:                5. 3048 mm

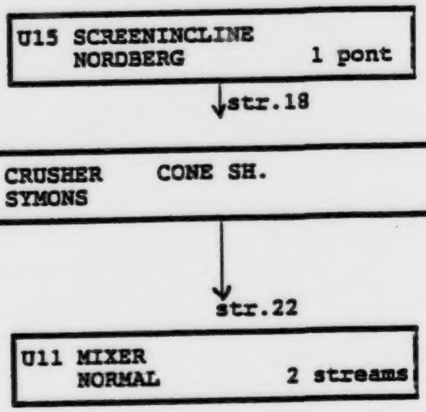
 Width:                 6. 1524 mm

 Wire opening:          7.   22 mm

 Wire diameter:         8.  10.000 mm
  (0 = slotted)

 Deck efficiency:     100.0 %
 G)ranulomytrie - <ret>continue  [ ]
```

FIGURE 24


U15 SCREENINCLINE
NORDBERG
1 pont
str.18
U16 CRUSHER
CONE SH.
SYMONS
str.22
U11 MIXER
NORMAL
2 streams

```
D)escription - G)ranulomytrie - U)nit  - <ret>continue  [ ]
```

```
                   str. 18          str. 22
610    mm         100.0 %          100.0 %
305    mm         100.0 %          100.0 %
203    mm         100.0 %          100.0 %
152    mm         100.0 %          100.0 %
102    mm         100.0 %          100.0 %
 89    mm         100.0 %          100.0 %
 76    mm         100.0 %          100.0 %
 63    mm          95.4 %          100.0 %
 51    mm          80.3 %          100.0 %
 38    mm          49.2 %          100.0 %
 25    mm          12.5 %          100.0 %
 19    mm           1.9 %          100.0 %
 12.7 mm             .1 %          100.0 %
  9.5 mm             .0 %          100.0 %
   4 mesh            .0 %           65.6 %
   8 mesh            .0 %           24.7 %
  16 mesh            .0 %           12.0 %
  30 mesh            .0 %            7.3 %
  50 mesh            .0 %            5.1 %
 200 mesh            .0 %            3.5 %
  Tph               130  tonnes      130  tonnes
  Water               0  tonne         0  tonne
Gr. max.             73 mm             9 mm
 D)escription - <ret> continue :
```

FIGURE 25

```
                        DESCRIPTION of unit no.     16
                                        - CRUSHER    CONE SH.

          1. Model:   SYMONS

          2. Capacity:    300  tonnes

          3. Maximum opening:        73 mm

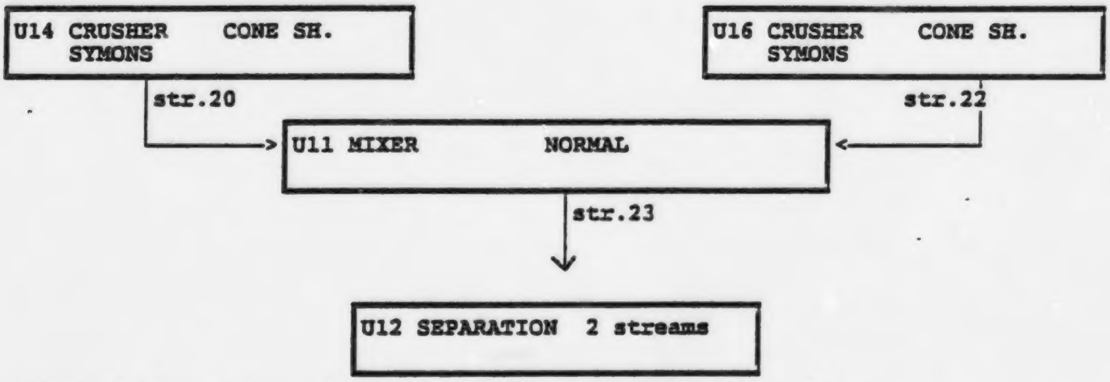
          4. Setting:     6 mm

          5. Cavity:  2  (1.extra fine, 2.fine, 3.medi
                           4."coarse", 5.extra "coarse")

             Efficiency :  100.0 %
             R/Reduction:    5.1
             Energy consumption:        104.1 kwh

G)ranulomytrie - <ret>continue  [ ]
```

FIGURE 26

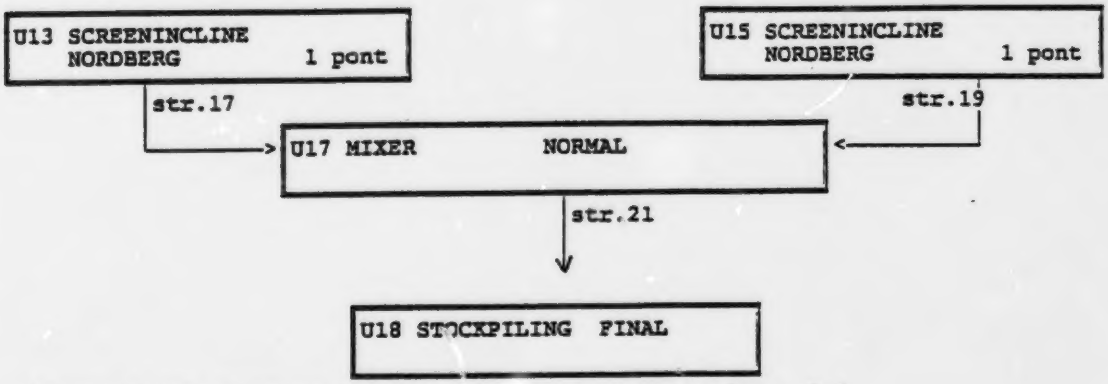

U14 CRUSHER CONE SH.
SYMONS
U16 CRUSHER CONE SH.
SYMONS
str.20
str.22
U11 MIXER NORMAL
str.23
U12 SEPARATION 2 streams

```
D)escription - G)ranulomytrie - U)nit  - <ret>continue  [ ]




                       str. 20          str. 22          str. 23
610    mm          100.0 %          100.0 %          100.0 %
305    mm          100.0 %          100.0 %          100.0 %
203    mm          100.0 %          100.0 %          100.0 %
152    mm          100.0 %          100.0 %          100.0 %
102    mm          100.0 %          100.0 %          100.0 %
 89    mm          100.0 %          100.0 %          100.0 %
 76    mm          100.0 %          100.0 %          100.0 %
 63    mm          100.0 %          100.0 %          100.0 %
 51    mm          100.0 %          100.0 %          100.0 %
 38    mm          100.0 %          100.0 %          100.0 %
 25    mm          100.0 %          100.0 %          100.0 %
 19    mm          100.0 %          100.0 %          100.0 %
 12.7 mm           100.0 %          100.0 %          100.0 %
  9.5 mm           100.0 %          100.0 %          100.0 %
  4 mesh            65.6 %           65.6 %           65.6 %
  8 mesh            24.7 %           24.7 %           24.7 %
 16 mesh            12.0 %           12.0 %           12.0 %
 30 mesh             7.3 %            7.3 %            7.3 %
 50 mesh             5.1 %            5.1 %            5.1 %
200 mesh             3.5 %            3.5 %            3.5 %
  Tph                130   tonnes     130   tonnes     260   tonnes
  Water                0   tonne        0   tonne        0   tonne
Gr. max.               9 mm             9 mm             9 mm
 D)escription - <ret> continue :
```

FIGURE 27


U13 SCREENINCLINE
NORDBERG 1 pont
U15 SCREENINCLINE
NORDBERG 1 pont
str.17
str.19
U17 MIXER NORMAL
str.21
U18 STOCKPILING FINAL

```
D)escription - G)ranulomytrie - U)nit  - <ret>continue  [ ]
```

```
                    str. 17          str. 19          str. 21
610    mm          100.0 %          100.0 %          100.0 %
305    mm          100.0 %          100.0 %          100.0 %
203    mm          100.0 %          100.0 %          100.0 %
152    mm          100.0 %          100.0 %          100.0 %
102    mm          100.0 %          100.0 %          100.0 %
 89    mm          100.0 %          100.0 %          100.0 %
 76    mm          100.0 %          100.0 %          100.0 %
 63    mm          100.0 %          100.0 %          100.0 %
 51    mm          100.0 %          100.0 %          100.0 %
 38    mm          100.0 %          100.0 %          100.0 %
 25    mm           99.3 %           99.3 %           99.3 %
 19    mm           95.1 %           95.1 %           95.1 %
 12.7 mm            88.7 %           88.7 %           88.7 %
  9.5 mm            85.8 %           85.8 %           85.8 %
  4 mesh            59.7 %           59.7 %           59.7 %
  8 mesh            28.0 %           28.0 %           28.0 %
 16 mesh            16.8 %           16.8 %           16.8 %
 30 mesh            11.7 %           11.7 %           11.7 %
 50 mesh             8.7 %            8.7 %            8.7 %
200 mesh             6.4 %            6.4 %            6.4 %
  Tph                 202   tonnes     202   tonnes     403   tonnes
  Water                 9   tonnes       9   tonnes      17   tonnes
Gr. max.               38  mm           38  mm           38  mm
 D)escription - <ret> continue :
```

FIGURE 28

```
                                                              11/ 7/88
                    CRUSHER SIMULATION PACKAGE


                     USER IDENTIFICATION

                    1. Name:  BRUNSWICK MINING

                    2. Reference number:   3

                    3. Date:  11/ 7/88



      SELECTION:   <ret>continue  - (#)correction - E)nd simulation
```

```
                       +-------------------+
                       |     Feed data     |
                       +-------------------+

  1. Type of Material      :  Ore

  2. Condition of Material:  WET

  3. Density of Material  :    4.300    tonnes / cubic metre

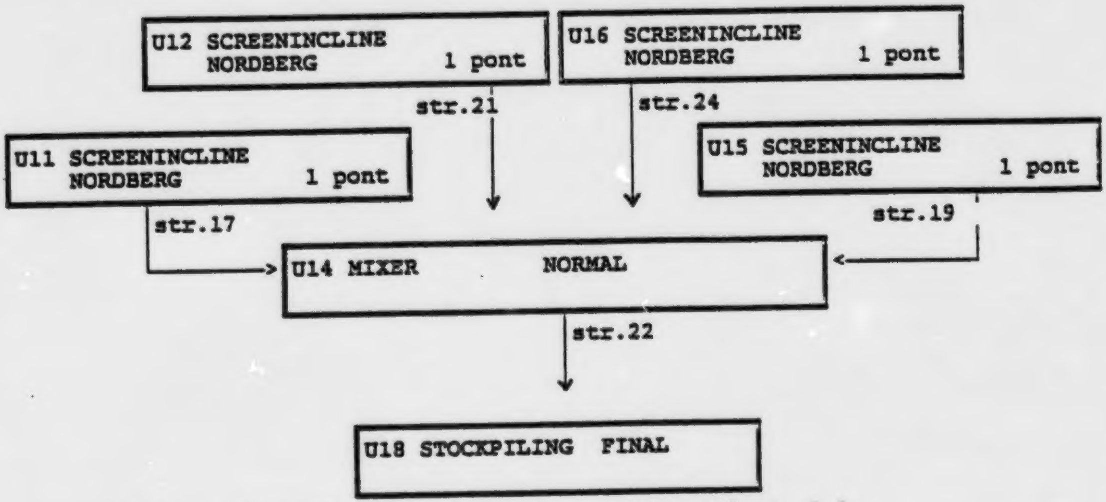
  4. Bond Index (WI)  :       12.500  Kwh / tonne

  5. % of Humidity (H2O)       4.0 %

  6. Tph (metric) to process :   420  tonnes / hour


          SELECTION: [ ]   <ret>continue   - (#)correction
```

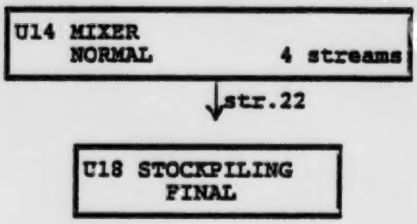
FIGURE 29


U12 SCREENINCLINE
NORDBERG
1 pont
U16 SCREENINCLINE
NORDBERG
1 pont
str.21
str.24
U11 SCREENINCLINE
NORDBERG
1 pont
U15 SCREENINCLINE
NORDBERG
1 pont
str.17
str.19
U14 MIXER
NORMAL
str.22
U18 STOCKPILING FINAL

```
D)escription - G)ranulomytrie - U)nit  - <ret>continue  [ ]
```

```
              str. 17    str. 19    str. 21    str. 24    str. 22
610      mm  100.0 %    100.0 %    100.0 %    100.0 %    100.0 %
305      mm  100.0 %    100.0 %    100.0 %    100.0 %    100.0 %
203      mm  100.0 %    100.0 %    100.0 %    100.0 %    100.0 %
152      mm  100.0 %    100.0 %    100.0 %    100.0 %    100.0 %
102      mm  100.0 %    100.0 %    100.0 %    100.0 %    100.0 %
 89      mm  100.0 %    100.0 %    100.0 %    100.0 %    100.0 %
 76      mm  100.0 %    100.0 %    100.0 %    100.0 %    100.0 %
 63      mm  100.0 %    100.0 %    100.0 %    100.0 %    100.0 %
 51      mm  100.0 %    100.0 %    100.0 %    100.0 %    100.0 %
 38      mm  100.0 %    100.0 %    100.0 %    100.0 %    100.0 %
 25      mm   99.3 %     99.3 %    100.0 %    100.0 %     99.3 %
 19      mm   95.0 %     95.0 %    100.0 %    100.0 %     95.0 %
 12.7    mm   88.6 %     88.6 %     27.9 %     27.9 %     88.4 %
  9.5    mm   85.7 %     85.7 %     15.4 %     15.4 %     85.5 %
  4    mesh   59.6 %     59.6 %       .0 %       .0 %     59.4 %
  8    mesh   28.0 %     28.0 %       .0 %       .0 %     27.8 %
 16    mesh   16.8 %     16.8 %       .0 %       .0 %     16.8 %
 30    mesh   11.7 %     11.7 %       .0 %       .0 %     11.7 %
 50    mesh    8.7 %      8.7 %       .0 %       .0 %      8.7 %
200    mesh    6.4 %      6.4 %       .0 %       .0 %      6.4 %
  Tph          201        201          1          1        403   tonnes
  Water          9          9          0          0         17   tonnes
Gr. max.        38 mm      38 mm      19 mm      19 mm      38 mm
 D)escription - <ret> continue :
```

FIGURE 30

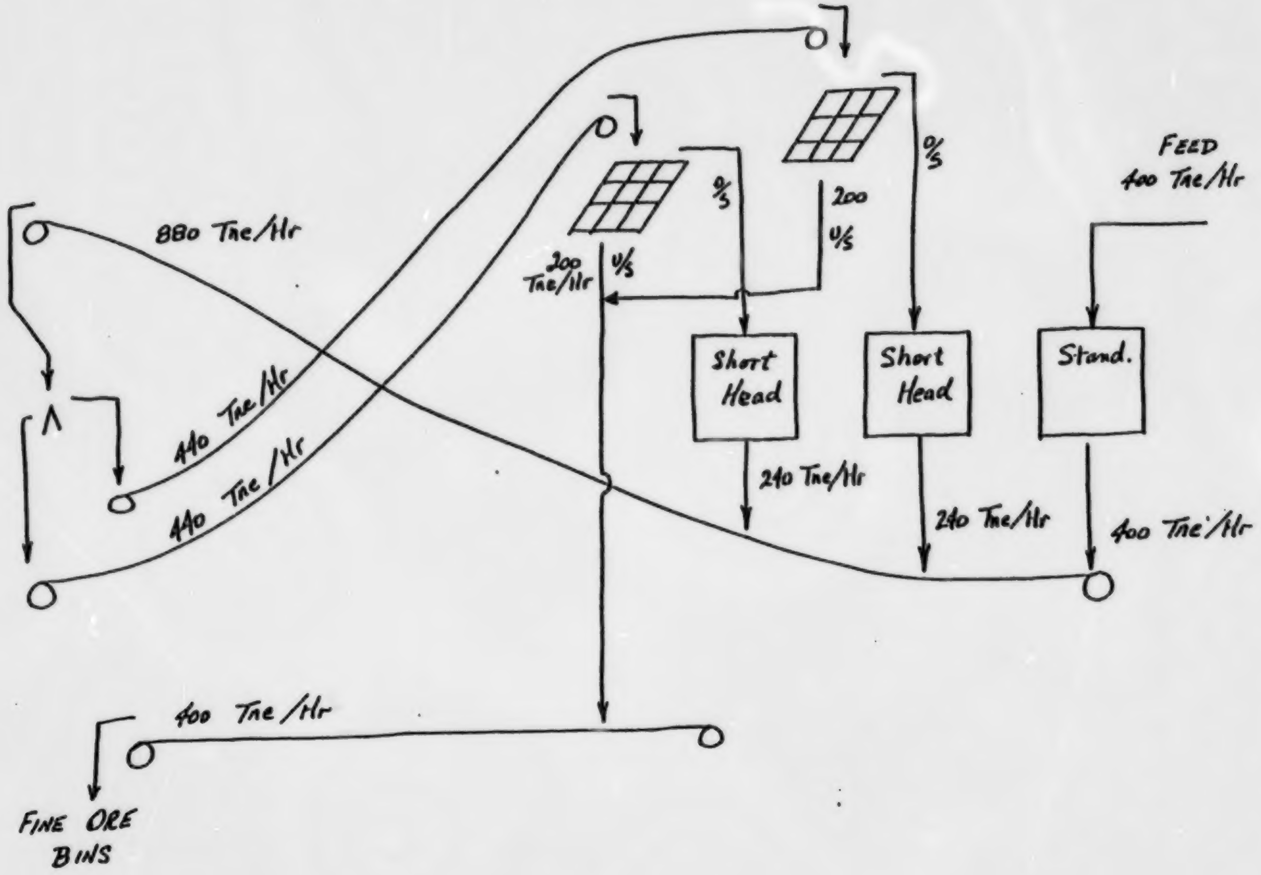

U14 MIXER
NORMAL
4 streams
str.22
U18 STOCKPILING
FINAL

```
D)escription - G)ranulomytrie - U)nit .- <ret>continue  [ ]
```

```
                      str. 22
 610     mm           100.0 %
 305     mm           100.0 %
 203     mm           100.0 %
 152     mm           100.0 %
 102     mm           100.0 %
  89     mm           100.0 %
  76     mm           100.0 %
  63     mm           100.0 %
  51     mm           100.0 %
  38     mm           100.0 %
  25     mm            99.3 %
  19     mm            95.0 %
  12.7   mm            88.4 %
   9.5   mm            85.5 %
   4 mesh              59.4 %
   8 mesh              27.8 %
  16 mesh              16.8 %
  30 mesh              11.7 %
  50 mesh               8.7 %
 200 mesh               6.4 %
   Tph                 403   tonnes
   Water                17   tonnes
 Gr. max.               38 mm
  D)escription - <ret> continue :
```

FIGURE 31

CRUSHER (SURFACE) FLOWSHEET


FEED
400 Tne/Hr
Stand.
400 Tne/Hr
Short Head
240 Tne/Hr
Short Head
240 Tne/Hr
O/S
O/S
200
U/S
200 Tne/Hr
U/S
880 Tne/Hr
440 Tne/Hr
440 Tne/Hr
400 Tne/Hr
FINE ORE BINS

* Both Crusher Plants are identical, operating tonnages shown.

FIGURE 32

UNDERGROUND CRUSHING:

1. GRIZZLY - None
2. JAW CRUSHER - 2950 Level
    - a. Type: 48" x 60" TRAYLOR
    - b. Capacity: 550 t/hr
    - c. Maximum Opening: 9"
    - d. Setting: 4½"
3. GYRATORY CRUSHER - 3000 Level (3900 Level 1989 Same Specs)
    - a. Type: 42" x 65" ALLIS CHALMERS
    - b. Capacity: 2000 t/hr
    - c. Maximum Opening: 8"
    - d. Setting: 3"

PRODUCT SCREEN ANALYSES - Not Available

FIGURE 33

SURFACE CRUSHING:

1. SECONDARY CRUSHER - One in each of two crushing plants

    a. Type: SYMONS 5½' Standard
    b. Capacity: 430 t/hr
    c. Maximum Opening: 14½"
    d. Setting: 1½"

2. TERTIARY CRUSHER - Two in each of two crushing plants

    a. Type: SYMONS 5½' Shorthead
    b. Capacity: 180 t/hr
    c. Maximum Opening: 3½"
    d. Setting: ¼"

3. SCREEN - Two for each of four shorthead crushers

    a. Type: NORDBERG 5 x 10
    b. Capacity: 15° for 5' from feed end
    5° for 5' at discharge end
    c. Length: 10'
    d. Width: 5'
    e. Screen: 3/4" thick urethane panels
    5' wide by 2' long each punched
    with 252 holes 1 1/8" diameter

    Partway through conversion to
    Polideck panels - 50 panels 1' x 1'
    each with 49 1" x 1" square holes,
    ½" thick

FIGURE 34

4. PRODUCT SCREEN ANALYSIS

WEIGHT %

SIZE	SCREEN FEED	SCREEN O/S	SCREEN V/S
2"	8.08	13.92	0
2½"	12.08	20.82	0
1	14.62	25.19	0
3/4"	16.37	25.55	3.68
1/2"	13.52	9.20	19.50
3/8"	11.58	2.07	24.74
3#	5.38	0.22	12.53
4#	3.58	0.07	8.42
6#	2.27	0.04	5.36
8#	1.76	0.03	4.17
12#	1.08	0.03	2.52
-12#	9.66	2.85	19.08

Screen Eff: 100 - % minus 3/4" in oversize

Screen efficiency of original screens ~80% with ~120% circulating load

Screen efficiency of Polideck panels ~95% with ~75% circulating load

APPENDIX 5

Calculation of Kinetic Rate Parameters

PROCESS STUDIES GROUP	technical notes	Project No.	File No.
	Project Flotation Simulation	Date 81-08-08	Page 1

Subject: Calculation of Kinetic Rate Parameters

Author(s): Donald S. Taylor

Objective: To compare three methods of calculation of kinetic rate parameters to determine if the data from the methods would give the same circuit design when used in the CALFLOT program. The three methods are:

1. The mathematical-graphical method supplied by Laval University.
2. The mathematical-graphical method using cell concentrations taken from "Mineral and Coal Flotation Circuits" by A. J. Lynch, *et. al*.
3. The computer program supplied by Laval University called CALIFLOT.

Conclusions: The data from the three calculation methods produces circuit designs that are similar. The kinetic rate parameters and the metallurgical performance of each calculation method are shown in Table I.

Author(s)	technical notes	Project No.	Page	of
D. Taylor			2	3

Discussion: There are several methods available to produce kinetic rate data from laboratory flotation tests. The first method (Attachment I) was a mathematical-graphical solution supplied by Laval University. The second method (Attachment II) uses a mathematical-graphical procedure involving cell concentrations to produce the kinetic rate data.

The third method is a computer program which uses mathematical algorithms to produce solutions for kinetic rate data. The program called CALIFLOT was supplied by Laval University and is able to obtain kinetic rate data very quickly and with greater accuracy than the mathematical-graphical methods. The program has the option to put in the relative accuracy of the output data. The lower the accuracy (only 1% error), the longer the calculation takes; however, the kinetic rate data are also more accurate. Therefore, we are running data at 5% and 1% accuracy error and obtaining similar results (Attachment III).

The kinetic rate data for each method are shown in Table 1, and although the data values are different, the results for circuit design in CALFLOT are very similar.

Author(s) D. Taylor	***technical notes***	*Project No.*	*Page* 3 *of* 3

Method: A laboratory test was done on antimony ore from Dominion Explorers according to standard procedures. The results of the test were run on a metallurgical balance (as shown in Attachment IV). The data was then analysed by the three different methods given in Attachments I, II and III, to produce the kinetic rate values. The kinetic rate values were then run on the CALFLOT computer program to produce the metallurgical data for the circuits (see Attachments V to VIII). The results from CALFLOT and the kinetic rate data were tabulated in Table I.

TABLE I

Data for Antimony
Test 2

METHOD	KINETIC RATE DATA				RESULTS FROM CALFLOT		
	Kf	Ks	Ø	R∞	Mineral Grade	Antimony Grade	Recovery
Mathematical-Graphical Laval University	0.505	0.051	0.192	0.774	63.5	45.53	73.2
Mathematical-Graphical Cell Concentrations	0.836	0.059	0.225	0.774	63.1	45.24	73.3
Computer Program CALIFLOT at 5% Estimated Error	0.795	0.066	0.214	0.765	64.7	46.39	72.8
Computer Program CALIFLOT at 1% Estimated Error	0.806	0.073	0.215	0.759	64.7	46.39	72.7

ATTACHMENT I

SUBJECT: Mathematical-Graphical Method for Calculating Kinetic Rate Parameters

OBJECTIVE: To produce kinetic rate parameters from standard rate kinetic flotation tests using a mathematical-graphical method supplied by Laval University in conjunction with the CALFLOT program. The method will produce Kf (fast floating rate constant), Ks (slow floating rate constant), Ø (the amount of slow floating mineral), and R_∞ (the maximum possible mineral recovery in the feed). This data can then be used by CALFLOT for flotation circuit design.

CONCLUSION: Through a series of mathematical and graphical solutions, we are able to obtain the required rate parameters.

DISCUSSION: The method of calculation shown here was supplied by Laval University in connection with our purchase of the CALFLOT program. The method has minor problems in obtaining accurate results due to interpolation of graphs used in the solution of the required constants. The method, however, agrees reasonably well with other methods of calculation.

METHOD: This mathematical-graphical method uses equations supplied by Mr. M. Bourassa of Laval University and is shown in Supplement I. The method used the equations shown in models 5, 6 and 7.

The first step in solving the rate kinetic problem is to determine the maximum recovery parameters-R_∞. This is accomplished by plotting a graph of recovery at time "T" versus recovery at time T + Δt. In this case, we chose a Δt the same as the original test design flotation times. The data is recorded in the table below:

TIME IN MINUTES	RECOVERY AT TIME	RECOVERY AT TIME + Δt
4	0.519	0.6017
8	0.6017	0.6667
16	0.6667	0.7127
24	0.7127	0.7293

The data is then plotted, as shown in Figure 1. The intersection of the line of the data points and a line with a slope of 1 going through the origin will give an R_∞ value of approximately 77.4% or $R_\infty = 0.774$.

Once R_∞ has been determined, we use the model "6", as shown in Supplement I, and use the relationship

$$\ln\left(1 - \frac{R}{R_\infty}\right) = \ln \O - Ks \cdot t$$

if R_∞ is known, and R then a plot of $\ln\left(1 - \frac{R}{R_\infty}\right)$

versus time will give the intercept Ø, and -Ks as the slope of the line. The data is recorded in the table below:

TIME IN MINUTES	RECOVERY	$\ln\left(1 - \frac{R}{R_\infty}\right)$
0.5	0.1852	not used
2.0	0.5190	not used
4.0	0.6017	not used
8.0	0.6667	-1.976
16.0	0.7127	-2.536
24.0	0.7293	-2.852

In this case, only the last three data points are used because we are looking for the slow floating material, and this will only occur at long flotation times.

The data was plotted on a graph, as shown in Figure 2, and from the graph, the Y-intercept is -1.65, therefore,

$$\O = e^{-1.65} = 0.192$$

and the slope of the line is $\frac{-17.5 - (-2.875)}{2 - 24} = -0.051$

therefore, $Ks = 0.051$.

Now using the second part of model 6 in Supplement I, we know Ks and Ø, the equation is:

$$\ln\left[\frac{1-\frac{R}{R\infty}-\phi\left(1-e^{-kst}\right)}{1-\phi}\right]=-kft$$

By plotting on a graph:

$$\ln\left[\frac{1-\frac{R}{R\infty}-\phi\left(1-e^{-kst}\right)}{1-\phi}\right]$$

versus time, as shown in Figure 3, we can determine from the slope of this line -Kf

$$-Kf = \frac{-0.371 - (-4.41)}{0 - 8} = -0.505$$

Kf = 0.505

This completes the calculation of the rate constants, which are summarized below:

Kf = 0.505
Ks = 0.051
Ø = 0.192
R∞ = 0.774

SUPPLEMENT I

FACULTE DES SCIENCES ET DE GENIE

THESE
PRESENTEE
A L'ECOLE DES GRADUES
DE L'UNIVERSITE LAVAL
POUR L'OBTENTION
DU GRADE DE MAITRE ES SCIENCES (M.Sc.)

PAR

MICHEL BOURASSA
BACHELIER ES SCIENCES APPLIQUEES
DE L'UNIVERSITE LAVAL

ETUDE LABORATOIRE-USINE DE LA FLOTTATION DE LA CHALCOPYRITE A LAC MATTAGAMI

OCTOBRE 1986

Reprinted with authorization of the author:

Michel Bourassa, P. Eng.

ANNEXE 4

METHODES D'ESTIMATION DES PARAMETRES DES MODELES CINETIQUES

Pour chacun des modèles on présente une méthode simple pour estimer les paramètres. Dans certains cas il existe d'autres méthodes pour évaluer les paramètres mais celles décrites ont l'avantage d'être des méthodes graphiques facilement utilisables.

Modèle 1: $R=1-(\lambda/\lambda+t)^{p}$

En posant $Y=1-R$ et en prenant le logarithme on a:

$$\ln(Y) = p\cdot\ln(\lambda) - p\cdot\ln(\lambda+p)$$

Lorsque le temps est suffisamment grand ($t>4-5\lambda$), $\ln(\lambda+t)\approx\ln(t)$. On a alors une relation linéaire en traçant Y en fonction de t sur papier logarithmique.

Relation linéaire : $\ln(Y) = p\cdot\ln(\lambda) - p\cdot\ln(t)$
ordonnée $= p\cdot\ln(\lambda)$
pente $= -p$

Pour améliorer la qualité de l'estimation, on répète la procédure en remplaçant t par $t+\lambda$ avec λ obtenu à l'itération précédente. La convergence est très rapide et deux itérations suffisent pour avoir de bons estimés. D'autres méthodes sont présenté par Inoue et Imaizumi[10] mais sont plus hasardeuses que celle-ci.

Modèle 2: $R=R_{\infty}[1-(\lambda/\lambda+t)^{p}]$

Les paramètres de ce modèle sont difficile à évaluer en raison de la présence du R_{∞}. Connaissant ce paramètre on trouve les autres paramètres de la même façon que pour le modèle 1. Pour évaluer R_{∞} on peut soit utiliser le R_{∞} d'un autre modèle ou y aller par tatonnement. On fixe alors R_{∞} pour calculer λ et la méthode d'estimation est la même que le modèle 1 soit:

Relation lin/aire : $\ln(1-[R/R_\infty] = p \cdot \ln(\lambda) - p \cdot \ln(t)$

ordonnée = $p \cdot \ln(\lambda)$

pente = $-p$

Modèle 3: $R=R_\infty(1-[(1-\exp-k \cdot t)/k \cdot t]\}$

Lorsque le temps est suffisamment grand (>3-4 min.), le terme exponentiel devient négligeable et la relation obtenue est linéaire. En traçant le graphe de R en fonction de 1/t on trouve les paramètres en se servant des coefficients de la droite.

Relation linéaire: $R=R_\infty - R_\infty/k \cdot t$

ordonnée = R_∞

pente = $-R_\infty/k$

Modèle 4: $R=R_\infty \cdot t/k+t$

Ce modèle se linéarise facilement en traçant le graphe de 1/R en fonction de 1/t

Relation linéaire : $1/R = 1/R_\infty + k/R_\infty \cdot t$

ordonnée = $1/R_\infty$

pente = k/R_∞

Modèle 5 : $R=R_\infty(1-\exp-k \cdot t)$

en posant $t_i=t_{i-1}+\Delta$ avec Δ=constante

on a: $R_i=R_\infty[1 - \exp(-k \cdot t_{i-1}) \cdot \exp(-k \cdot \Delta)]$

$R_i=R_\infty[1 - \xi \cdot \exp-k \cdot t_{i-1}]$ (1)

avec $\xi=\exp(-k \cdot \Delta)$=constante

de plus, $R_{i-1}=R_\infty(1 - \exp(-k \cdot t_{i-1})$

$\Longrightarrow \exp(-k \cdot t_{i-1}) = (R_\infty - R_{i-1})/R_\infty$ (2)

On trouve finalement, en remplaçant [1] dans [2], une relation linéaire entre R_t et R_{t-1}.

relation linéaire : $R_t=R_\infty(1-\xi) + \xi\cdot R_{t-1}$
ordonnée = $R_\infty(1-\xi)$
pente = ξ

Une fois qu'on connait R_∞, on trace sur une échelle semi-logarithmique $1-(R/R_\infty)$ en fonction du temps.

relation linéaire : $\ln[1 - (R/R_\infty)] = -k\cdot t$
ordonnée = 0
exp. pente = $-k$

Modèle 6: $R=(1-\phi)(1-\exp[-k_f\cdot t]) + \phi(1-\exp[-k_s\cdot t])$

Comme k_f est beaucoup plus grand que k_s (environ 10 fois), le terme $\exp(-k_f\cdot t)$ devient négligeable lorsque t est assez grand (t>3-4 min.). On obtient une relation linéaire en traçant 1-R en fonction de t sur une échelle semi- logarithmique.

Relation linéaire : $\ln(1-R)=\ln(\phi) - k_s\cdot t$ (t>3-4 min.)
ordonnée = $\ln(\phi)$
pente = $-k_s$

Lorsque l'on connait ϕ et k_s, on a une autre relation linéaire qui nous donne la valeur de k_f.

Relation linéaire : $\ln\{1- [R-\phi(1-\exp-k_s\cdot t)]/[1-\phi]\} = -k_f\cdot t$
ordonnée = 0
③ pente = $-k_f$

Modèle 7: $R=R_\infty\{(1-\phi)(1-\exp-k_f\cdot t)+\phi(1-\exp-k_s\cdot t)\}$

En faisant la même hypothèse que pour le modèle 6, lorsque t est grand le modèle devient:

$$R=R_\infty[1-\phi\cdot\exp(-k_s\cdot t)]$$

Le paramètre R_∞ se détermine de la même façon que pour celui du modèle 5. Une fois ce paramètre connu, le problème à résoudre est similaire à celui du modèle précédent (6).

$$R/R_\infty=(1-\phi)(1-\exp[-k_f\cdot t])+\phi(1-\exp[-k_s\cdot t])$$

FIGURE 1

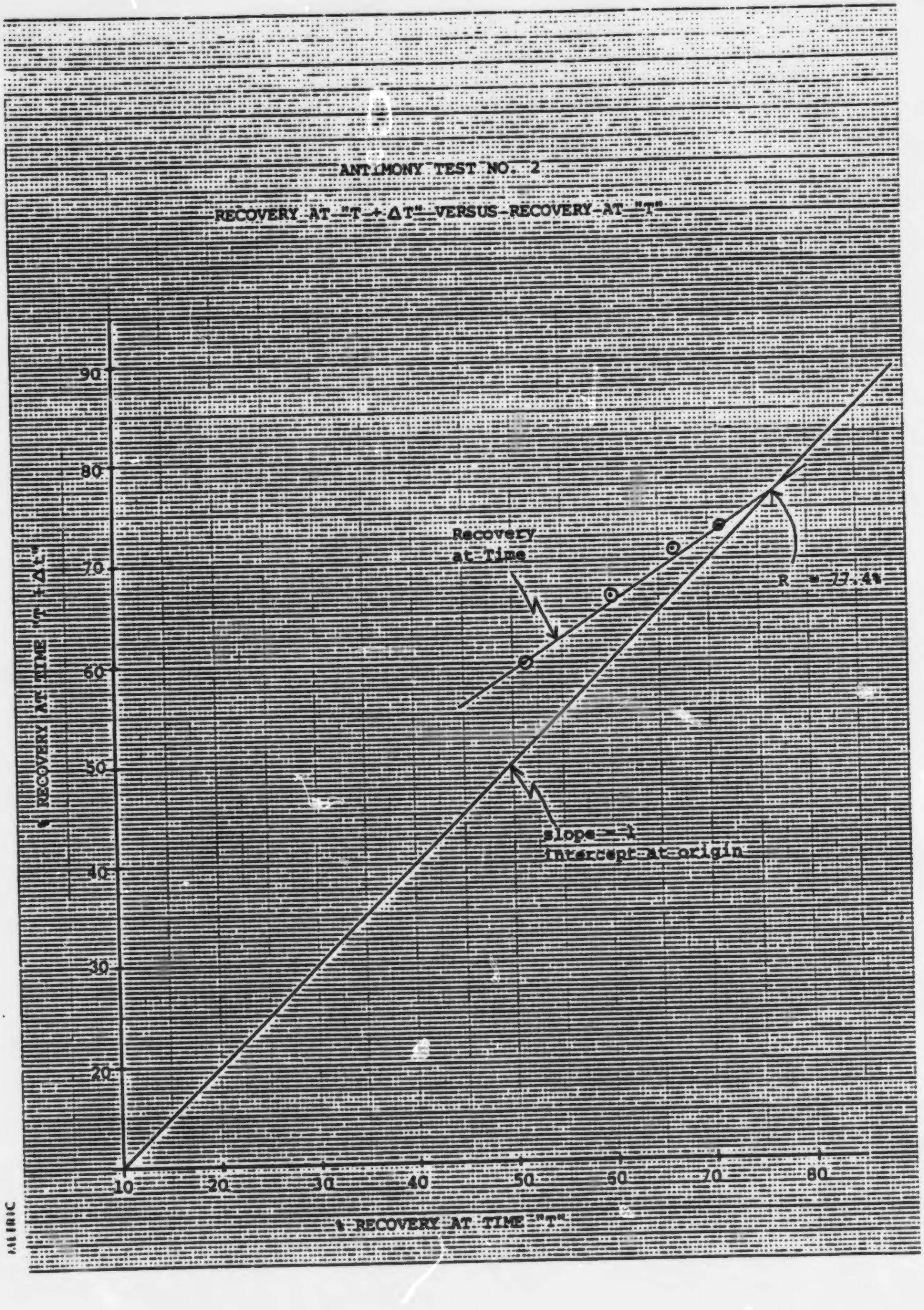

ANTIMONY TEST NO. 2
RECOVERY AT "T + ΔT" VERSUS RECOVERY AT "T"
% RECOVERY AT TIME "T + Δt"
90
80
70
60
50
40
30
20
10
20
30
40
50
60
70
80
% RECOVERY AT TIME "T"
Recovery at Time
R = 77.4%
slope = 1
intercept at origin
METRIC

FIGURE 2

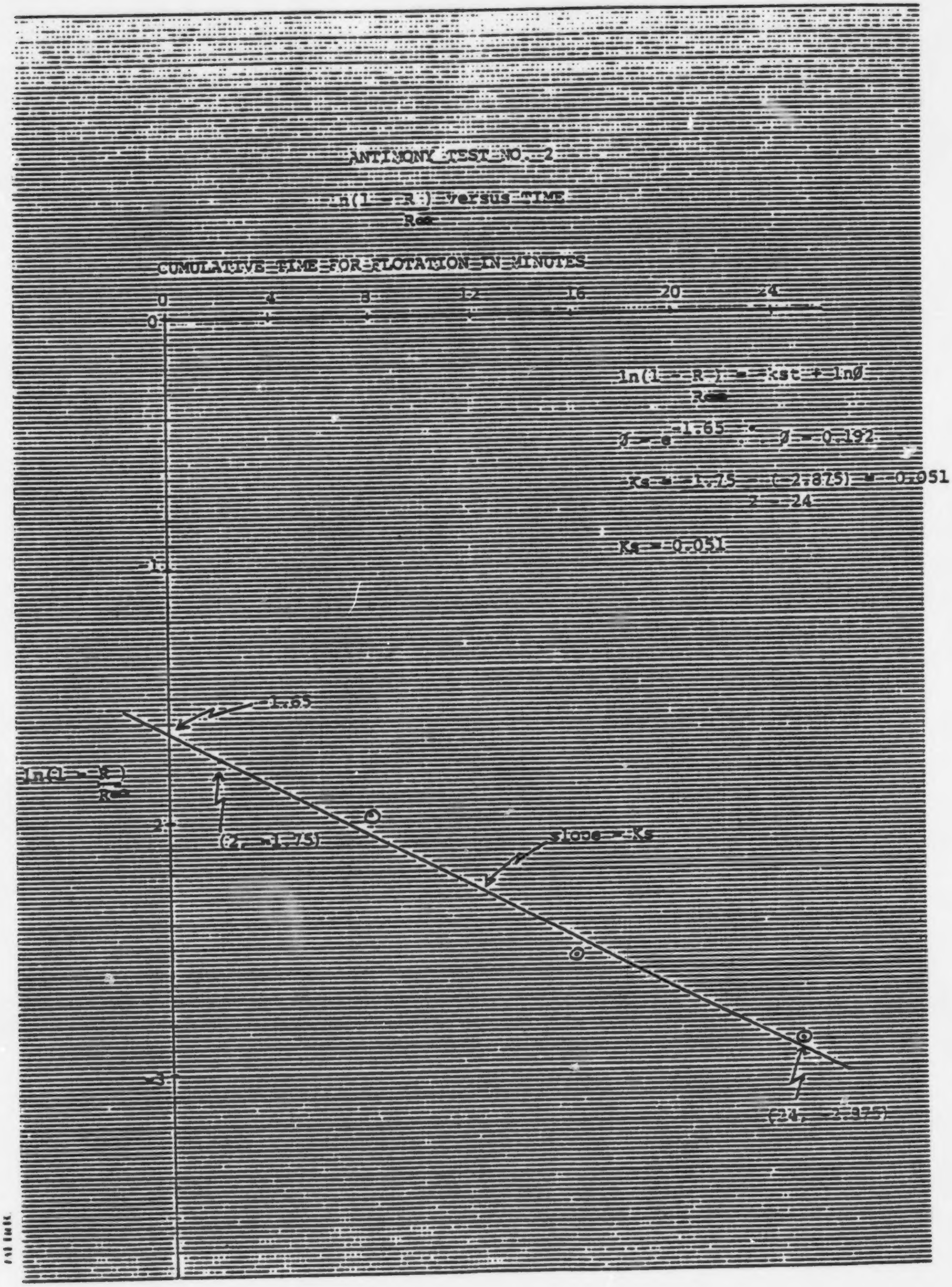
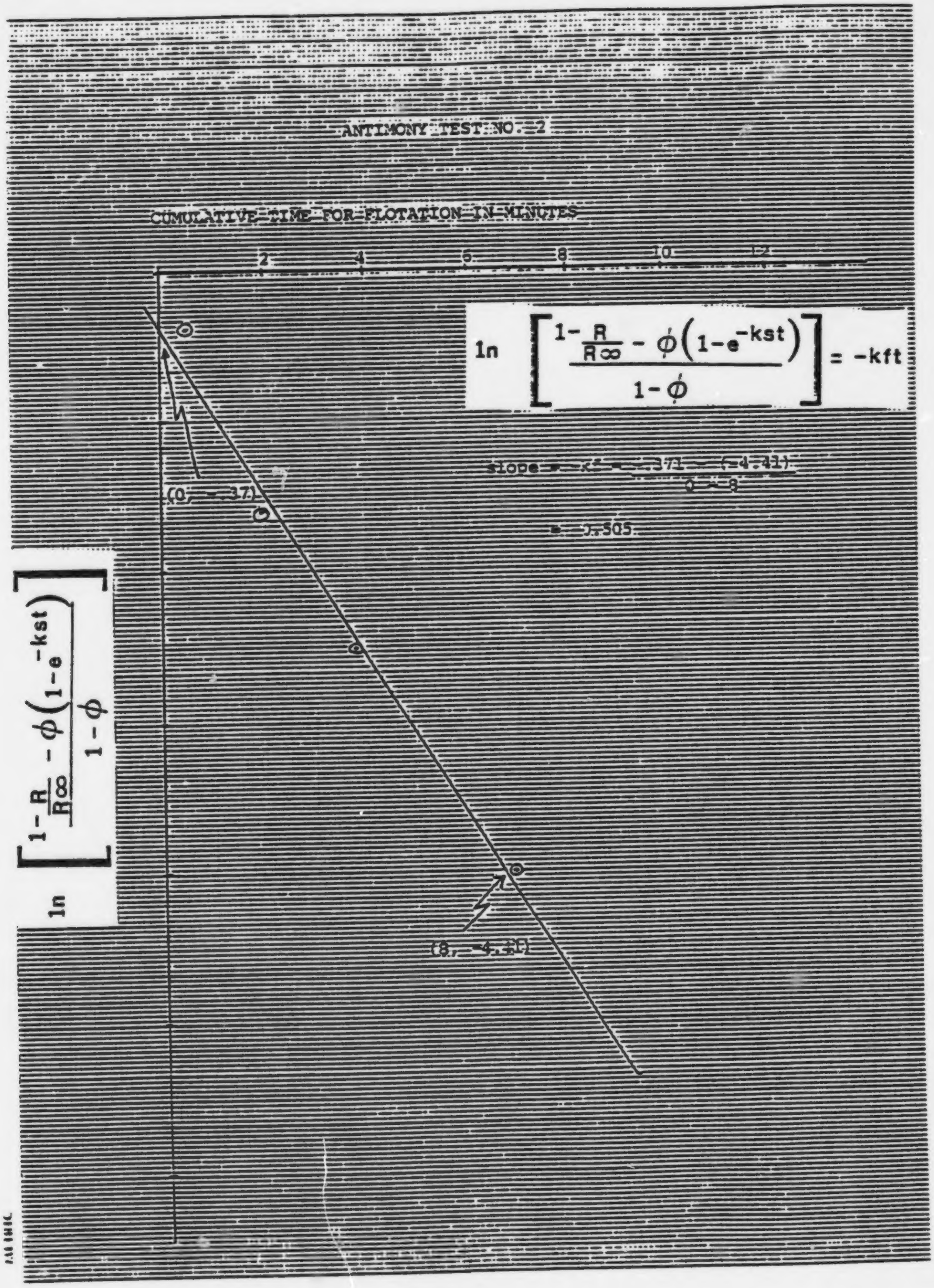

ANTIMONY TEST NO. 2
ln(1 - R/R∞) versus TIME
CUMULATIVE TIME FOR FLOTATION IN MINUTES
0
4
8
12
16
20
24
0
-1
-2
-3
ln(1 - R/R∞) = -kst + lnØ
Ø = e^-1.65 ∴ Ø = 0.192
Ks = -1.75 - (-2.875) / 2 - 24 = -0.051
Ks = 0.051
-1.65
ln(1 - R/R∞)
(2, -1.75)
slope = -Ks
(24, -2.875)

FIGURE 3


ANTIMONY TEST NO. 2
CUMULATIVE TIME FOR FLOTATION IN MINUTES
2
4
6
8
10
12
$\ln\left[\frac{1-\frac{R}{R\infty}-\phi\left(1-e^{-kst}\right)}{1-\phi}\right]=-kft$
slope = -kf = $\frac{-.371-(-4.41)}{0-8}$
= -0.505
(0, -.37)
(8, -4.41)
$\ln\left[\frac{1-\frac{R}{R\infty}-\phi\left(1-e^{-kst}\right)}{1-\phi}\right]$

ATTACHMENT II

SUBJECT: Cell Concentration Method for Calculating Kinetic Rate Parameters

OBJECTIVE: To produce kinetic rate parameters from standard rate kinetic flotation tests, using the method of cell concentrations, as shown in Supplement II. The method will produce Kf (fast floating rate constant), Ks (slow floating rate constant), Ø (the amount of slow floating mineral, and R^{∞} (the maximum possible mineral recovery in the feed). This data can then be used by CALFLOT for flotation circuit design.

CONCLUSION: Through the use of cell concentrations using a mathematical-graphical method, we are able to obtain the required rate parameters.

DISCUSSION: The method of calculation shown here was taken from the book "Mineral and Coal Flotation Circuits" by A.J. Lynch et. al published by Elsevier Scientific Publishing Company. This method was used to verify that the methods supplied by Laval University were reasonably accurate when calculating rate parameters. However, the method has some minor problems, the most sever being interpolation of graphs leading to errors in accurate results for the rate parameters. The method, however, agrees well with other methods of calculation.

METHOD: The mathematical-graphical method requires the initial concentration of the floatable species in the cell called "Co" to be calculated. In this case, we assume a maximum recovery of about 5% above the last test recovery, and in this case, the maximum recovery used is 77.4%. "Co" is then calculated on the basis of the free mineral species, in this case SbS, in which 71.7% of the weight is Sb.

As our assays are done on the basis of Sb, calculations are necessary to convert to the mineral species. For "Co" we take the feed assay (Figure 4) and calculate the weight of pure Sb = 3.46% Sb (head assay) x 77.4% (recovery maximum) x 990.0 g (weight of feed) = 26.58 g of Sb. This is now converted to SbS:

$$26.58 \times \frac{100}{71.7} = 37.06 \text{ g of Sbs.}$$

The amount of SbS removed at each test is calculated from the assay and weight of each concentrate sample taken during the test interval. At the time t = 0.5, the SbS is calculated by 10.48 g (weight of concentration) x 60.70% (assay of Sb) x $\frac{100}{71.7}$ (Sb to SbS) = 8.87 g SbS (see Figure 5 for data). This amount of SbS is subtracted from the original amount of SbS to give a new concentration in the cell of SbS. When all the data for each time interval is completed, then

the ratio of $\frac{C(t)}{Co} = \frac{\text{concentration at time (t)}}{\text{concentration at time (o)}}$ is found

The results are shown in Figure 5

When the ratio of $\frac{C(t)}{Co}$ has been calculated a graph of

$\frac{C(t)}{Co}$ is plotted versus time (Figure 6).

An estimate of Cs (concentration for slow floating mineral) is made by taking the last four data points and drawing a straight line through them to the Y-axis. From this line Cs the value of Cs is taken, as shown in Figure 6.

In Supplement I, the equation 3.16 is shown as
$C = Co \cdot [\emptyset \cdot \exp(-Ks \cdot t) + (1 - \emptyset) \cdot \exp(-Kf \cdot t)]$

in this equation: $Cs = \emptyset \cdot \exp(-Ks \cdot t)$
$Cf = (1 - \emptyset) \cdot \exp(-Kf \cdot t)$

Therefore, $\frac{C}{Co} = Cs + Cf$, and if we have

Cs from the graph and $\frac{C(t)}{Co}$, then Cf can

be found by subtraction and is shown in Figure 6. Note that only the first three data points are used, for that is only where the fast floating species would occur.

The rate kinetic terms can now be worked out.

(1) $\emptyset$ is where Cs crosses the Y-axis and in this case it is 0.225.

(2) The fast float rate constant Kf

$Cf = (1 - \phi) \cdot \exp(-Kf \cdot t)$, using $\phi = 0.225$ and the first three time intervals:

Time	Kf
t = 0.5	0.7244
t = 2.0	0.9560
t = 4.0	0.8270

Average = 0.836 = Kf

(3) The slow float rate constant Ks

$Cs = \phi \cdot \exp(-Ks \cdot t)$ using $\phi = 0.225$ and the last three time intervals:

Time	Kf
8.0	0.0402
16.0	0.0497
24.0	0.0858

Average = 0.0586 = Ks

(4) Data Summary

Kf = 0.836
Ks = 0.059
ϕ = 0.229
$R\infty$ = 0.774

The data is then run on CALFLOT to give a circuit design, as shown in Attachment VI.

SUPPLEMENT II

5.4 CALCULATION OF MODEL PARAMETERS

Procedures for the calculation of parameters in flotation models will be explained through numerical examples. There are many parameters which have been used in various models and only those which are regarded as important in circuit optimisation and control will be considered in the examples.

5.4.1 *Flotation rate constants*

The simple two rate constant model given in eq. 3.16 describes with reasonable accuracy the behaviour of the valuable particles in many semi-

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Title: Mineral and Coal Flotation Circuits - Their Simulation and Control

Series: Developments in Mineral Processing, Volume 3

Authors: A. J. Lynch, N.W. Johnson, E. V., Manlapig, C. G. Thorne

Pages: 125-128, 65-67

batch processes. For a continuous cell eq. 3.25 is the basis of the model

$$R_s = K_s \cdot \tau / (1 + K_s \cdot \tau) \; ; R_f = K_f \cdot \tau / (1 + K_f \cdot \tau) \; ; R = \phi \cdot R_s + (1 - \phi) \cdot R$$

The derivation of the parameters K_f, K_s and ϕ from plant data is given in Example 5-2.

Example 5-2: A sampling survey has been carried out on a rougher bank in which galena is floated and the following data are derived about bank performance from the completed mass balances.

Cell No.	1	2	3	4	5	6	7	8	9	10	11	12
C/C_0	0.556	0.312	0.201	0.135	0.098	0.083	0.069	0.055	0.051	0.044	0.039	0.035

The data are shown in Fig. 5-5. It may be assumed that the retention time is 20 seconds per cell. The problem is to calculate the parameters K_f, K_s and ϕ for the galena. The following sequence of calculations are carried out to solve the problem.

(1) K_s is estimated by assuming that only slow floating particles float in the last four cells. For these cells

$$R = (C_8 - C_{12}) / C_8 = 1 - (1 + K_s \cdot \tau)^{-N}$$

$$0.3636 = 1 - (1 + K_s \cdot 0.33)^{-4}$$

$$K_s = 0.362 \text{ min}^{(-1)}$$

(2) The value of ϕ, which is the proportion of galena which has a low rate constant, is estimated either graphically by projecting the straight line connecting the concentrations at cells 8 and 12 on the concentration-cell number curve in Fig. 5-5 to intersect the ordinate, or algebraically as follows:

$$R = (\phi - C) / \phi = 1 - (1 + K_s \cdot \tau)^{-N}$$

For the slow floating fraction after 8 cells or 2.67 minutes retention time

$$(\phi - 0.055) / \phi = 1 - (1 + \underset{0.362}{\cancel{0.33}} \cdot \underset{0.33}{\tau})^{-8}$$

$$\phi = 0.1362$$

(3) The behaviour of the low rate constant fraction can be determined on a cell-by-cell basis. The equation

$$R_N = (\phi - C_N) / \phi = 1 - (1 + K_s \cdot \tau)^{-N}$$

The values at $C_0 - C_{12}$ are

(0.1362), 0.1218, 0.1086, 0.0970, 0.0867, 0.0773, 0.0680, 0.0617, 0.0550, 0.0491, 0.0439, 0.0392, 0.0350.

(4) The behaviour of the high rate constant fraction may be determined on cell-by-cell basis by subtracting the low rate constant fraction for the total particles. The value for the cell 1 tailing is 0.5560 − 0.1218 = 0.4362 and the values for each cell are

(0.8638), 0.4342, 0.2034, 0.1040, 0.0483, 0.0207, 0.0140, 0.0073.

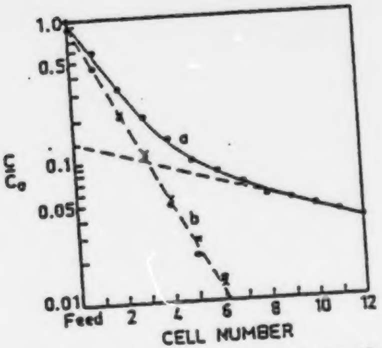


Fig. 5.5. Line a – observed fraction of galena in successive cells down a flotation bank.

Line b – observed and calculated behaviour of the fast-floating fraction after the behaviour of the slow-floating fraction is accounted for.

(5) K_f may now be calculated using the data from the first cell.

$$(0.8638 - 0.4342) / 0.8638 = 1 - [1 + 0.33 \cdot K_f]^{-1}$$

$K_f = 3.00 \text{ min}^{-1}$

A comparison of the observed and calculated values of the high rate constant fraction, using the calculated value of K_f, as it proceeds down the bank is as follows.

Cell No.	1	2	3	4	5	6	7
Observed fraction in tailing	0.4342	0.2034	0.1040	0.0483	0.0207	0.0140	0.0073
Calculated fraction in tailing	0.4342	0.2151	0.1074	0.0541	0.0268	0.0134	0.0067

The agreement is shown in Fig. 5–5.

The best estimates of the model parameters K_f, K_s and ϕ may be calculated using a standard least squares fitting procedure. This involves minimissing $\Sigma_i\ (0_i - C_i)$ with respect to the model parameters. 0_i and C_i are the observed and calculated values at the i^{th} time interval. Deming (1964) may be consulted for the mathematical techniques.

The calculation of model parameters must be carried out with full recognition of the conditions of process operation as these may be such as to cause complications. For instance inhibited flotation conditions and/or froth overloading may occur early in the rougher and cleaner banks in sulphide flotation and this means that there is an additional mechanism operating which restricts the rates at which particles appear in the concentrates irrespective of their true flotation rates. Use of the normal procedure for the calculation of flotation rates for cells in which the additional mechanism is operating will lead to erroneous results. Another complication may occur when collector is added during the flotation process. This may affect any or all of the three parameters and it is necessary to recognise that different values of the parameters may exist before and after collector addition. Similarly variation in water flow to the concentrate caused by the addition of frother or by variation in pulp level during the process may cause difficulty in interpretation of data.

The procedures for including changes in these variables in the model are discussed in chapter 6.

The factors affecting the flotation rate constants are discussed in more detail in chapter 6. It is important to emphasise, however, that in some cases the assumption that valuable particles may be classified into two groups with high and low rate constants respectively may be invalid and in these cases one of the more complex models discussed in chapter 3 must be used for simulation purposes. For instance if the ore is very fine grained the occurrence of composite particles with a range of rate constants must be recognised. Numerical examples in this monograph will be restricted to the two rate constant case as this case covers most industrial operations which have been investigated.

particles are deposited into the concentrate launder at a rate which is proportional to the concentration of those particles in the pulp (pulp to froth transfer being the rate determing step).

The simple rate equation for a semi-batch flotation process may be written

$$-dC/dt = K \cdot C \tag{3.12}$$

where C is the concentration of particles with identical flotation properties in the pulp and t is time. The rate constant K is related to the flotation conditions such as the collector concentration and pulp level. The term $(-dC/dt)$ represents the instantaneous recovery rate from a semi-batch compartment.

If the term C_0 represents the initial concentration of particles with identical flotation properties in the pulp of a semi-batch compartment, the integrated form of the rate equation is

$$C = C_0 \cdot \exp(-K \cdot t) \tag{3.13}$$

The similarity between this equation (for a semi-batch compartment) and the probability equation proposed by Kelsall (for a bank of continuous compartments at steady state), eq. 3.11, will be noted in the expanded form of eq. 3.13 which is

$$C = C_0 \cdot \left[1 - K + \frac{K^2}{2!} - \frac{K^3}{3!} + \ldots\right]^t \tag{3.14}$$

and which approximates to $C = C_0 \cdot (1 - K)^t$ since terms in K^2 and above become small. The parameters C and W, t and n, and K and P are synonymous. Both equations assume plug flow characteristics for the compartment in their development (see also Section 3.4.2).

If the particles in the pulp do not possess identical flotation properties but possess a continuous range of rate constants, the integrated form of the rate equation for a semi-batch compartment becomes

$$C = C_0 \cdot \int_0^\infty \exp(-K \cdot t)\, f(K, 0)\, dk \tag{3.15}$$

where $f(K, 0)$ represents a continuous distribution of rate constants.

The major problem with eq. 3.15 is related to the accurate estimation of the distribution of the rate constants.

With regard to this distribution, Kelsall (1961) suggested that the approximation could frequently be made that the floatable species could be divided into two components with high and low flotation rate constants. In the cases where this approximation is valid the rate equation for a semi-batch laboratory cell may be written

$$C = C_0 \cdot [\phi \cdot \exp(-K_s \cdot t) + (1 - \phi) \cdot \exp(-K_f \cdot t)] \tag{3.16}$$

where K_s and K_f are the two rate constants and ϕ is the proportion of the floatable species with low rate constant K_s.

The concepts embodied in eqs. 3.13 and 3.16 are illustrated in the following numerical examples.

Example 3.1. The progress of semi-batch flotation under ideal conditions can be described by the first order rate equation

$$C = C_0 \cdot \exp(-K \cdot t)$$

where C and C_0 are the concentrations of the floatable component in the cell at time t and zero time respectively, and K is the flotation rate constant. Calculate and plot the mass fraction of the component in the cell after each minute of flotation up to 8 minutes when K is 0.5 and 1.0 min^{-1}.

Solution

The mass fraction C/C_0 of the species in the cell as a function of time for different flotation rate constants is as follows

t	1	2	3	4	5	6	7	8
K = 1.0 min^{-1}	0.368	0.135	0.049	0.018	0.007	0.002	0.0009	0.0003
K = 0.5 min^{-1}	0.606	0.368	0.223	0.135	0.082	0.049	0.030	0.018

The results are shown graphically in Fig. 3–4.

Example 3.2. In example 3.1 a single floatable component only was considered but in many real processes it has been found that the approximation may be made that the valuables in the feed are made up of two components which are fast and slow floating respectively. The equation describing the progress of flotation of this feed may be written

$$C = C_0 \cdot [\phi \cdot \exp(-K_s \cdot t) + (1 - \phi) \cdot \exp(-K_f \cdot t)]$$

where K_s and K_f are the flotation rate constants of the slow and fast floating components and ϕ is the fraction of the valuable species which is slow floating. Calculate and plot the mass fraction of the valuable component in the cell after each minute of flotation up to 8 minutes when K_f and K_s are 1.0 and 0.1 min^{-1} respectively and ϕ is 0.2.

Solution

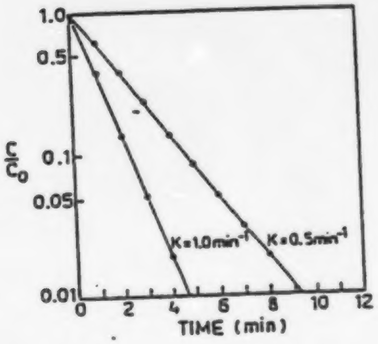
The mass fraction of the valuable component in the cell as a function of time may be calculated from the equation and the results are given in Table 3–I.

TABLE 3–I

Calculated concentration of the valuable component in the cell as a function of time when the component includes fast and slow floating species.

t (min)	$C_f = 0.8 \cdot e^{-1.0 \cdot t}$	$C_s = 0.2 \cdot e^{-0.1 \cdot t}$	$C/C_0 = C_f + C_s$
1	0.2943	0.1810	0.4753
2	0.1083	0.1637	0.2720
3	0.0398	0.1482	0.1880
4	0.0146	0.1341	0.1486
5	0.0054	0.1231	0.1267
6	0.0020	0.1098	0.1118
7	0.0007	0.0993	0.1000
8	0.0002	0.0899	0.0901

The results are shown graphically in Fig. 3–5.



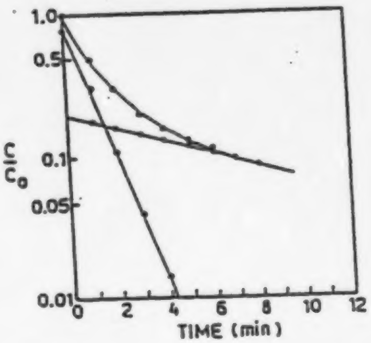


Fig. 3–4. Batch flotation with a single floatable species – effect of change in rate constant.

Fig. 3–5. Batch flotation with two floatable species having high and low rate constants.

3.4.2 *Residence time distribution in isolated continuous compartments*

The recovery of a mineral in a continuous compartment is governed not only by the distribution of rate constants describing particle floatabilities but also by the distribution of the retention times of particles in the pulp volume which depends on the type of mixing or flow regime (see Levenspiel, 1962, pp. 309–323). Thus for a single component with a distribution of residence times

$$C = C_0 \cdot \int_0^\infty \exp(-K \cdot t)\, E(t)\, dt \tag{3.17}$$

and for many components

$$C = C_0 \cdot \int_0^\infty \int_0^\infty \exp(-K \cdot t)\, E(t) f(K \cdot 0)\, dt\, dK \tag{3.18}$$

where terms C and C_0 can have units of concentration or flowrate. Therefore it is necessary to examine the type of mixing which exists in the pulp volume.

Danckwerts (1953) recognised two limiting conditions of mixing regimes in a continuous reactor. In one condition there exists a continuous gradation in the concentration of components over the length of reactor or isolated compartment. In such a reactor, all particles in the exit (tailing) stream have spent an equal time in the reactor (isolated compartment). In a semi-batch flotation compartment, all particles have also spent an equal time in the reactor. If all particles spend an equal time in a reactor, the reactor has piston, plug or tube flow mixing characteristics and the three terms are

FIGURE 4

GRADE AND RECOVERY

METAL: Sb SAMPLE NAME ANIMONY FLOAT II FILE NAME CALFLOT

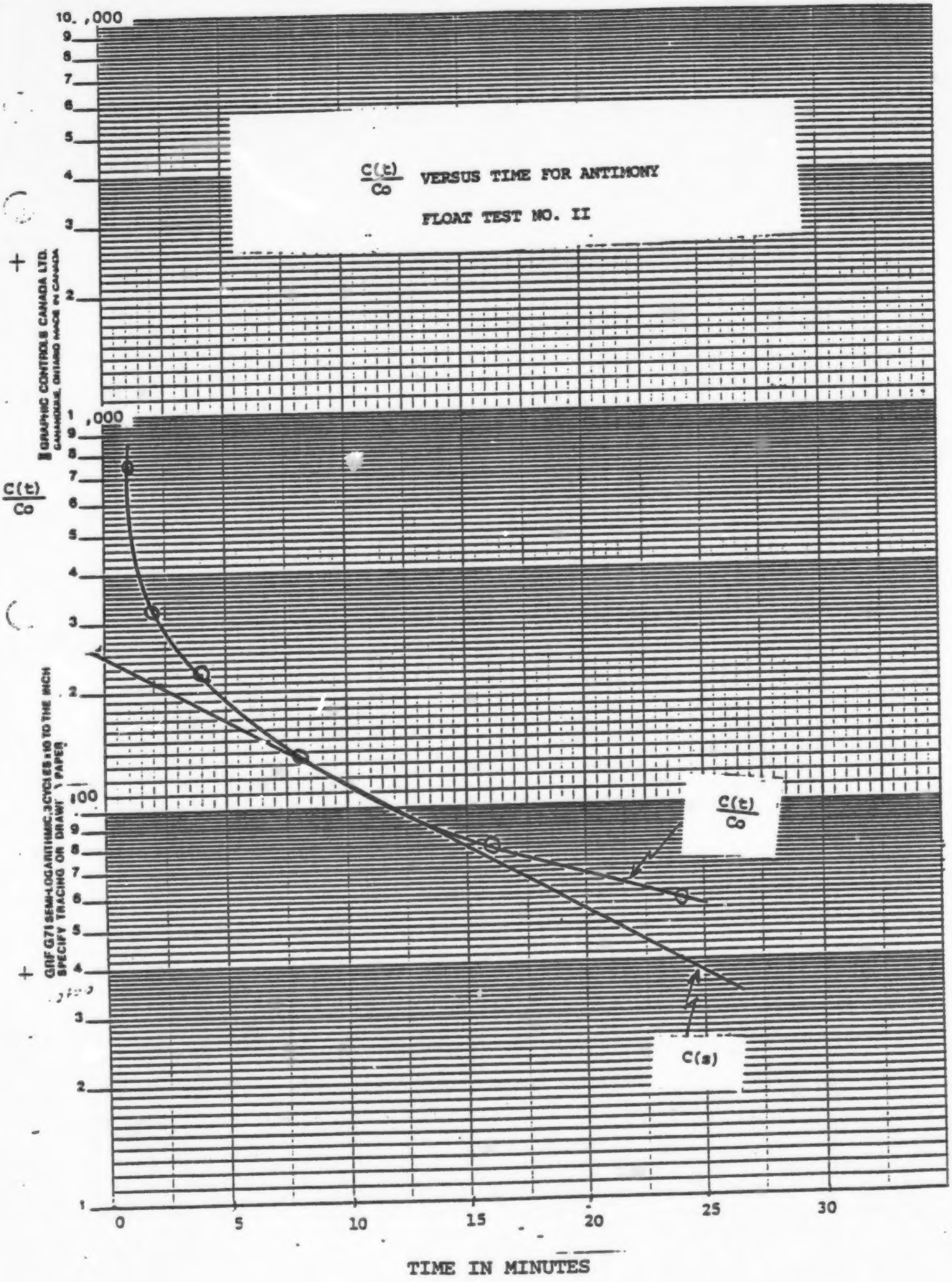
SAMPLE	WEIGHT (grams)	WEIGHT PERCENT	METAL ASSAY	METAL CONTENT	METAL DISTRIBUTION	GRADE	CUMULATIVE % WEIGHT	CUMULATIVE % RECOVERY
TAILS	929.82	93.92	1.00	93.92	27.07	41.62	6.08	72.93
24	5.45	0.55	10.40	5.73	1.65	0.00	5.53	71.27
16	10.20	1.03	15.50	15.97	4.60	0.00	4.50	66.67
8	7.94	0.80	28.10	22.54	6.50	0.00	3.70	60.17
4	6.24	0.63	45.50	28.68	8.27	0.00	3.07	51.91
2	19.87	2.01	57.70	115.81	33.38	0.00	1.06	18.52
.30	10.48	1.06	60.70	64.26	18.52	0.00	-0.00	0.00
N/A	0.00	0.00	0.00	0.00	0.00	0.00	-0.00	0.00
	0.00	0.00	0.00	0.00	0.00	0.00	-0.00	0.00
	0.00	0.00	0.00	0.00	0.00	0.00	-0.00	0.00
	0.00	0.00	0.00	0.00	0.00	0.00	-0.00	0.00
FEED	990.00		3.30	346.90				

BALANCE: 105.12

FIGURE 5

Cumulative Time in Minutes	Weight of SbS Removed in Concentrate	Weight of SbS Remaining in Flotation Cell	Ratio $\frac{C(t)}{Co}$	Cs Concentration Slow Float	Cf Concentration Fast Float
0	0.0	37.06	1.00		
0.5	8.87	28.19	0.7606	0.2209	0.5397
2.0	15.99	12.20	0.3238	0.2093	0.1145
4.0	3.96	8.24	0.2223	0.1939	0.0284
8.0	3.11	5.13	0.1384	0.1631	
16.0	2.20	2.93	0.0791	0.1015	
24.0	0.790	2.14	0.0577	0.0399	

FIGURE 6



$\frac{C(t)}{Co}$ VERSUS TIME FOR ANTIMONY

FLOAT TEST NO. II

ATTACHMENT III

```
Test Identification : ANTIMONY 2

  Cumulative Time          Cumulative Recovery
       (min)             measured        calculated
1       0.5               0.1852           .2025
2       2                 0.5191           .4989
3       4                 0.6017           .6142
4       8                 0.6667           .6673
5       16                0.7127           .7079
6       24                0.7293           .7313
7
8
9
10

Parameter      Start Value      Relative          Optimal Value
                                accuracy (%)
 Kf              1                 5                  .795
 Ks              0.1               5                  .066
 Phi             0.2               5                  .214
R inf.           0.75              5                  .765

        ROOT SQARE OF RESIDUES : 0.029879
```

```
Test Identification : ANTIMONY 2

  Cumulative Time          Cumulative Recovery
       (min)             measured        calculated
1       0.5               0.1852           .2034
2       2                 0.5191           .4991
3       4                 0.6017           .6136
4       8                 0.6667           .6674
5       16                0.7127           .7087
6       24                0.7293           .7311
7
8
9
10

Parameter      Start Value      Relative          Optimal Value
                                accuracy (%)
 Kf              1                 1                  .806
 Ks              0.1               1                  .073
 Phi             0.2               1                  .215
R inf.           0.75              1                  .759

        ROOT SQARE OF RESIDUES : 0.029864
```

ATTACHMENT IV

GRADE AND RECOVERY

METAL: Sb SAMPLE NAME ANIMONY FLOAT II FILE NAME CALFLOT

SAMPLE	WEIGHT (grams)	WEIGHT PERCENT	METAL ASSAY	METAL CONTENT	METAL DISTRIBUTION	GRADE	CUMULATIVE % WEIGHT	CUMULATIVE % RECOVERY
TAILS	929.82	93.92	1.00	93.92	27.07	41.62	6.08	72.93
24	5.45	0.55	10.40	5.73	1.65	0.00	5.53	71.27
16	10.20	1.03	15.50	15.97	4.60	0.00	4.50	66.67
8	7.94	0.80	28.10	22.54	6.50	0.00	3.70	60.17
4	6.24	0.63	45.50	28.68	8.27	0.00	3.07	51.91
2	19.87	2.01	57.70	115.81	33.38	0.00	1.06	18.52
.30	10.48	1.06	60.70	64.26	18.52	0.00	-0.00	0.00
N/A	0.00	0.00	0.00	0.00	0.00	0.00	-0.00	0.00
	0.00	0.00	0.00	0.00	0.00	0.00	-0.00	0.00
	0.00	0.00	0.00	0.00	0.00	0.00	-0.00	0.00
	0.00	0.00	0.00	0.00	0.00	0.00	-0.00	0.00
FEED	990.00		3.30	346.90				

BALANCE: 105.12

ATTACHMENT V

```
MATHEMATICAL-GRAPHICAL - LAVAL UNIVERSITY
  ****** PRINT : INPUT DATA ******


   CALCULATION MODE: Rougher-Scavenger Alone

   CIRCUIT CONFIGURATION
ROUGHER: Chose by Programm
                                                                RS
   DATA CONFIGURATION:
 Kinetic Parametres Known (R.inf.,Kf,Ks,Phi) ?                   Y
 Entrainment Coef. for Unfloat minerals Known (CF) ?             Y
 Rate Constants for Water known ?                                N
 Volume Cells fixed ?                                            N
 Mark of Cells fixed ?                                           N
 Number of Cells per Bank fixed ?                                N
 Number of Cells Fixed ?                                         N
 Water Flowrate for Froth Collapse known ?                       N
 Proportion of Slow Floating in mids known ?                     N
 Number of Cells for Scavenger fixed ?                           N
 Minimum Grade for Concentrate fixed ?                           N




 Solid Flowrate (T.M./h)=                  100


        Relative Density
     ANTIMONY     4.7
     QUARTZ       2.6



                     MINERAL FEED GRADE (frac.)

   ANTIMONY         .0466
   QUARTZ           .9534
                    ------
                    1.000



                     Entrainment Coefficients (CF)


                          QUARTZ       .14
```

	RGHR ANTIMO
Kf:Fast Rate(1/min)	.505
Scale-up Fact on Kf	1.000
Ks:Slow Rate(1/min)	.051
Scale-up Fact on Ks	1.000
Slow prop. (f)	.192
Slow Prop in mids	.192
Ultimate Rec. (f)	.774

	RGHR
Percent of Solids in Feed (f.)	.420
Water rate constant (1/min.)	.005

	RGHR
Maximum Recovery for Cells (f)	.300
Minimum Recovery Justified for a Cell (f)	.0050
Max Percent Solid for Concentrates (f)	.350
Tolerance for Residence Time Calcul. (f)	.0100
Tolerance for Entrainment Calculation (f)	.0100
Tolerance for Circulating Flow Calcul. (f)	.0100
Max Iterations for Residence Time Calcul.	1
Max Iterations for Entrainment Calculation	8
Max Iterations for Circulating Flow Calcul.	10

```
                    ****** PRINT : EQUIPEMENT ******


      CIRCUIT                      Rghr
Number of Paral. Banks:              1
Number of Cells/Bank  :             12
Volume of Cells (m3)  :            7.99
                (pi3) :            282.0
Type of Cells:                     simple
Power Installed (HP):              20.0
Manufacturer(s) * :                  0
Flotation Time (min):              26.2
Unit Flotation Time :              2.18
Alternative Volume(m3):            4.25
                                   5.66
                                   8.50


   *  MANUFACTURER: (W)Wemco, (A)Agitair, (O) Outokumpu, (D)Dorr
```

```
                   ****** RESULTS : SUMMARY BALANCE ******

                              CIRCUIT : ROUGHER

                                        FLOWRATE t/h
                      New Feed      Rougher Conc         Tail
Solid                 100.000          5.372            94.628
Water                 138.095         18.950           119.145
Water Addition                         2.467
Water adjusment                         .000

                               GRADE  and DISTRIBUTION %
                       New Feed        Rougher Conc          Tail
                    Grade     Dist    Grade      Dist    Grade     Dist
ANTIMONY              4.7    100.0     63.5      73.2      1.3     26.8
QUARTZ               95.3    100.0     36.5       2.1     98.7     97.9
Water                58.0    100.0     77.9      13.7     55.7     86.3
Solid                42.0    100.0     22.1       5.4     44.3     94.6
```

ATTACHMENT VI

```
              MATHEMATICAL-GRAPHICAL - CELL CONCENTRATIONS
                  ****** PRINT : INPUT DATA ******


    CALCULATION MODE: Rougher-Scavenger Alone

    CIRCUIT CONFIGURATION
ROUGHER: Chose by Programm
                                                                RS
    DATA CONFIGURATION:
  Kinetic Parametres Known (R.inf.,Kf,Ks,Phi) ?                  Y
  Entrainment Coef. for Unfloat minerals Known (CF) ?            Y
  Rate Constants for Water known ?                               N
  Volume Cells fixed ?                                           N
  Mark of Cells fixed ?                                          N
  Number of Cells per Bank fixed ?                               N
  Number of Cells Fixed ?                                        N
  Water Flowrate for Froth Collapse known ?                      N
  Proportion of Slow Floating in mids known ?                    N
  Number of Cells for Scavenger fixed ?                          N
  Minimum Grade for Concentrate fixed ?



  Solid Flowrate (T.M./h)=                     100


           Relative Density
        ANTIMONY     4.7
        QUARTZ       2.6


                        MINERAL FEED GRADE (frac.)

     ANTIMONY          .0466
     QUARTZ            .9534
                       ------
                       1.000


                        Entrainment Coefficients (CF)


                             QUARTZ        .14
```

	RGHR ANTIMO
Kf:Fast Rate(l/min)	.836
Scale-up Fact on Kf	1.000
Ks:Slow Rate(l/min)	.059
Scale-up Fact on Ks	1.000
Slow prop. (f)	.225
Slow Prop in mids	.225
Ultimate Rec. (f)	.774

	RGHR
Percent of Solids in Feed (f.)	.420
Water rate constant (l/min.)	.005

	RGHR
Maximum Recovery for Cells (f)	.300
Minimum Recovery Justified for a Cell (f)	.0050
Max Percent Solid for Concentrates (f)	.350
Tolerance for Residence Time Calcul. (f)	.0100
Tolerance for Entrainment Calculation (f)	.0100
Tolerance for Circulating Flow Calcul. (f)	.0100
Max Iterations for Residence Time Calcul.	1
Max Iterations for Entrainment Calculation	8
Max Iterations for Circulating Flow Calcul.	10

```
                  ****** PRINT : EQUIPEMENT ******


     CIRCUIT                     Rghr
Number of Paral. Banks:            1
Number of Cells/Bank  :           12
Volume of Cells (m3)  :          7.99
                (pi3) :          282.0
Type of Cells:                  simple
Power Installed (HP):            20.0
Manufacturer(s) * :               0
Flotation Time (min):            26.2
Unit Flotation Time :            2.18
Alternative Volume(m3):          4.25
                                 5.66
                                 8.50


   *  MANUFACTURER: (W)Wemco, (A)Agitair, (O) Outokumpu, (D)Dorr
```

```
                 ****** RESULTS : SUMMARY BALANCE ******

                           CIRCUIT : ROUGHER

                                     FLOWRATE t/h
                  New Feed      Rougher Conc         Tail
Solid            100.000            5.416           94.584
Water            138.095           19.292          118.803
Water Addition                      2.493
Water adjusment                      .000

                           GRADE  and DISTRIBUTION %
                  New Feed      Rougher Conc         Tail
               Grade    Dist   Grade    Dist    Grade    Dist
ANTIMONY         4.7   100.0    63.1    73.3      1.3    26.7
QUARTZ          95.3   100.0    36.9     2.1     98.7    97.9
Water           58.0   100.0    78.1    14.0     55.7    86.0
Solid           42.0   100.0    21.9     5.4     44.3    94.6
```

ATTACHMENT VII
LAVAL CALIFLOT COMPUTER PROGRAM - 5% RELATIVE ACCURACY

```
****** PRINT : INPUT DATA ******

   CALCULATION MODE: Rougher-Scavenger Alone

   CIRCUIT CONFIGURATION
ROUGHER: Chose by Programm
                                                                RS
   DATA CONFIGURATION:
 Kinetic Parametres Known (R.inf.,Kf,Ks,Phi) ?                   Y
 Entrainment Coef. for Unfloat minerals Known (CF) ?             Y
 Rate Constants for Water known ?                                N
 Volume Cells fixed ?                                            N
 Mark of Cells fixed ?                                           N
 Number of Cells per Bank fixed ?                                N
 Number of Cells Fixed ?                                         N
 Water Flowrate for Froth Collapse known ?                       N
 Proportion of Slow Floating in mids known ?                     N
 Number of Cells for Scavenger fixed ?                           N
 Minimum Grade for Concentrate fixed ?                           N



 Solid Flowrate (T.M./h)=                   100


          Relative Density
        ANTIMONY      4.7
        QUARTZ        2.6


                         MINERAL FEED GRADE (frac.)

     ANTIMONY          .0466
     QUARTZ            .9534
                       ------
                       1.000


                         Entrainment Coefficients (CF)


                             QUARTZ        .14
```

```
                          RGHR
                          ANTIMO

Kf:Fast Rate(l/min)        .795
Scale-up Fact on Kf       1.000
Ks:Slow Rate(l/min)        .066
Scale-up Fact on Ks       1.000
Slow prop. (f)             .214
Slow Prop in mids          .214
Ultimate Rec. (f)          .765
```

```
                                             RGHR
     Percent of Solids in Feed (f.)          .420
     Water rate constant (l/min.)            .005
```

```
                                                  RGHR
Maximum Recovery for Cells (f)                    .300
Minimum Recovery Justified for a Cell (f)        .0050
Max Percent Solid for Concentrates (f)            .350
Tolerance for Residence Time Calcul. (f)         .0100
Tolerance for Entrainment Calculation (f)        .0100
Tolerance for Circulating Flow Calcul. (f)       .0100
Max Iterations for Residence Time Calcul.            1
Max Iterations for Entrainment Calculation           8
Max Iterations for Circulating Flow Calcul.         10
```

```
                    ****** PRINT : EQUIPEMENT ******


    CIRCUIT                      Rghr
Number of Paral. Banks:            1
Number of Cells/Bank  :           11
Volume of Cells (m3)  :          7.99
                (pi3) :         282.0
Type of Cells:                  simple
Power Installed (HP):            20.0
Manufacturer(s) * :               0
Flotation Time (min):            23.9
Unit Flotation Time :            2.17
Alternative Volume(m3):          4.25
                                 5.66
                                 8.50


  *  MANUFACTURER: (W)Wemco, (A)Agitair, (O) Outokumpu, (D)Dorr
```

```
                 ****** RESULTS : SUMMARY BALANCE ******

                          CIRCUIT : ROUGHER

                                   FLOWRATE t/h
                      New Feed        Rougher Conc         Tail
Solid                  100.000            5.237           94.763
Water                  138.095           17.907          120.188
Water Addition                            2.386
Water adjusment                            .000

                               GRADE  and DISTRIBUTION %
                      New Feed        Rougher Conc           Tail
                    Grade     Dist     Grade      Dist     Grade     Dist
ANTIMONY              4.7    100.0      64.7      72.8       1.3     27.2
QUARTZ               95.3    100.0      35.3       1.9      98.7     98.1
Water                58.0    100.0      77.4      13.0      55.9     87.0
Solid                42.0    100.0      22.6       5.2      44.1     94.8
```

ATTACHMENT VIII

LAVAL CALIFLOT COMPUTER PROGRAM _ 1% RELATIVE ACCURACY

```
                    ****** PRINT : INPUT DATA ******


    CALCULATION MODE: Rougher-Scavenger Alone

    CIRCUIT CONFIGURATION
ROUGHER: Chose by Programm
                                                                RS
     DATA CONFIGURATION:
  Kinetic Parametres Known (R.inf.,Kf,Ks,Phi) ?                  Y
  Entrainment Coef. for Unfloat minerals Known (CF) ?            Y
  Rate Constants for Water known ?                               N
  Volume Cells fixed ?                                           N
  Mark of Cells fixed ?                                          N
  Number of Cells per Bank fixed ?                               N
  Number of Cells Fixed ?                                        N
  Water Flowrate for Froth Collapse known ?                      N
  Proportion of Slow Floating in mids known ?                    N
  Number of Cells for Scavenger fixed ?                          N
  Minimum Grade for Concentrate fixed ?                          N



  Solid Flowrate (T.M./h)=                    100


          Relative Density
       ANTIMONY     4.7
       QUARTZ       2.6


                        MINERAL FEED GRADE (frac.)


    ANTIMONY           .0466
    QUARTZ             .9534
                       ------
                       1.000


                       Entrainment Coefficients (CF)


                          QUARTZ       .14
```

	RGHR ANTIMO
Kf:Fast Rate(l/min)	.806
Scale-up Fact on Kf	1.000
Ks:Slow Rate(l/min)	.073
Scale-up Fact on Ks	1.000
Slow prop. (f)	.215
Slow Prop in mids	.215
Ultimate Rec. (f)	.759

	RGHR
Percent of Solids in Feed (f.)	.420
Water rate constant (l/min.)	.005

	RGHR
Maximum Recovery for Cells (f)	.300
Minimum Recovery Justified for a Cell (f)	.0050
Max Percent Solid for Concentrates (f)	.350
Tolerance for Residence Time Calcul. (f)	.0100
Tolerance for Entrainment Calculation (f)	.0100
Tolerance for Circulating Flow Calcul. (f)	.0100
Max Iterations for Residence Time Calcul.	1
Max Iterations for Entrainment Calculation	8
Max Iterations for Circulating Flow Calcul.	10

```
                 ****** PRINT : EQUIPEMENT ******

      CIRCUIT                  Rghr
Number of Paral. Banks:          1
Number of Cells/Bank  :         11
Volume of Cells (m3)  :        7.99
                (pi3) :        282.0
Type of Cells:                simple
Power Installed (HP):          20.0
Manufacturer(s) * :             O
Flotation Time (min):          23.9
Unit Flotation Time :          2.17
Alternative Volume(m3):        4.25
                               5.66
                               8.50


   *  MANUFACTURER: (W)Wemco, (A)Agitair, (O) Outokumpu, (D)Dorr
```

```
                   ****** RESULTS : SUMMARY BALANCE ******

                            CIRCUIT : ROUGHER

                                      FLOWRATE t/h
                      New Feed     Rougher Conc        Tail
Solid                 100.000          5.233          94.767
Water                 138.095         17.907         120.188
Water Addition                         2.385
Water adjusment                         .000

                             GRADE  and DISTRIBUTION %
                     New Feed       Rougher Conc        Tail
                  Grade     Dist    Grade     Dist    Grade     Dist
ANTIMONY            4.7    100.0     64.7     72.7      1.3     27.3
QUARTZ             95.3    100.0     35.3      1.9     98.7     98.1
Water              58.0    100.0     77.4     13.0     55.9     87.0
Solid              42.0    100.0     22.6      5.2     44.1     94.8
```

APPENDIX 6

The CALFLOT Flotation Design Simulator and the CALIFLOT Kinetic Rate Constant Calculator

PROCESS STUDIES GROUP	*technical notes*	*Project No.*	*File No.*
	Project Flotation Simulation	*Date* 07/09/88	*Page* 1

Subject: The CALFLOT flotation design simulator, and the CALIFLOT kinetic rate constant calculator.

Author(s): Donald S. Taylor

Objective: In the past few years, plant simulators have been introduced into mineral processing operations to help engineers improve plant production and design new plants. As a part of the Department of Natural Resources and Energy, we have obtained these simulators to help the province's processing plants and to aid the department in producing preliminary flow sheets for plant design and costing.

The programs for the unit operation "Flotation" were obtained from Laval University, the G.R.A.I.I.M. group of the Department of Mining and Metallurgy. The programs will be described along with the data and the laboratory procedures needed to obtain data for the simulator.

Conclusions: The following simulators were made operable on our computer system:

1. CALFLOT program, which gives circuit design flow rates and metallurgical recoveries.
2. CALIFLOT program, which calculates the kinetic rate data from laboratory flotation tests to be used in the CALFLOT program.

By standard mathematical methods, the following data can be calculated:

1. Kinetic rate data from laboratory tests using a mathematical-graphical method supplied by Laval University.
2. Kinetic rate data from laboratory tests involving a mathematical-graphical method using the concentration of floatable material in the cell (called cell concentration).
3. Calculate Entrainment Coefficients which measure the "tails solids", carried by water in the froth to the concentrate product.

Author(s) D. Taylor	technical notes	Project No.	Page of 2

Discussion: The simulation of flotation circuits has been the subject of many technical papers in the last 10 years. The objectives of the simulators are:

1. to design a flotation circuit by running a series of laboratory flotation tests on a new orebody or zone without pilot plant operation.
2. to obtain samples in an operating plant circuit, run tests in a laboratory cell, then simulate the circuit on the computer program to see if improvements can be made by modifying or rearranging the flotation circuit.

In searching for a program to fulfil this need, we were able to obtain from Laval University a program, which they have been using for the last few years, on simulation of flotation circuits. The program called CALFLOT is able to use flotation kinetic rate data obtained from laboratory flotation tests to design a flotation circuit. The program will design the circuit configuration, give a list of recommended equipment, and show flow rates and metal distributions around the flotation circuits.

The most essential information needed to operate the CALFLOT program is the "kinetic rate data". The kinetic rate is the measured rate at which floatable particles move from the flotation cell to the concentrate product. Researchers have found that two rates are required; one for fast floating particles and one for slow floating particles.

The conversion of laboratory flotation test data to kinetic rate data is very difficult, and requires, first, that the laboratory tests be done under very strict conditions for time and water control. The second problem is to convert the flotation data to kinetic rate data using mathematical methods. Several attempts were successfully made at converting data using mathematical- graphical methods. The first method, supplied by Laval University, is shown in Attachment I and a second method, using cell concentrations, is shown in Attachment II. The difficulty with these methods lies in the interpolation of the graphs used in the conversion methods. The accuracy of this kinetic data depends on the "eye" of the interpreter in handling the graphical data.

The engineers at Laval University have produced a computer program called CALIFLOT, which uses mathematical algorithms and reiterative calculations to produce kinetic rate data from flotation tests. The computer program uses statistical methods to eliminate the graphical interpretation of data. An example of the output of the program is shown in Attachment III.

Author(s) D. Taylor	***technical notes***	*Project No.*	*Page* *of* 3

Discussion: A second input needed for the computer program CALFLOT is the data on "entrainment coefficients". The entrainment coefficient is defined as the non-floatable waste rock which is carried by the water in the froth to the concentrate product. It is the result of solids being carried by entrapment in the froth water of the concentrate product. A method for calculation of this entrainment coefficient is shown in Attachment IV.

The test work carried out on these programs was done on antimony ore from Dominion Explorers because of its availability, the simple ore product flotation concentrate, and the previous flotation work carried out on the ore.

Author(s) D. Taylor	***technical notes***	*Project No.*	*Page of* 4

Method: The flotation tests were carried out on a standard Denver laboratory flotation machine. A sample cell was chosen for the appropriate feed weight. The sample was then ground in the laboratory rod mill until the desired liberation grind was produced. The ground sample was placed on the machine, the impeller placed in the cell, and then water was added to bring up the cell level. This added water was also weighed. The cell was started, the reagents added, then the air was turned on for 30 seconds for the first concentrate float. The cell was shut down, the amount of wash water used was weighed, and the amount of make-up water needed to bring back to its normal level was weighed. This sequence was repeated a second time for 1.5 minutes, then a 2 minute, a 4 minute, and two 8 minute floats were done (See Attachment V). After the last float, the cell was once again brought back to level, the water weights were recorded, and then all components (concentrates and tail) were filtered, dried, weighed and sent for antimony assay.

The assays and weights were then run on a standard metallurgical balance program to produce time recovery data (shown in Attachment VI). The data from the metallurgical balance was then treated by the mathematical-graphical methods shown in Attachments I and II, or on the CALIFLOT program shown in Attachment III. The kinetic rate data methods produced the parameters Ks (fast floating rate constant), Kf (slow floating rate constant), Ø (the amount of slow floating mineral), and R∞ (the maximum possible mineral recovery in the feed).

The CALIFLOT program, besides giving the kinetic data, provides an estimate of the accuracy of its computer calculations. At the end of each run, the computer calculates the "Root Square of Residues", which is a measure of the difference between the measured and calculated recovery. A number of 0.010 to 0.030 shows a very good fit of data to the mathematical solution.

The calculations for the entrainment coefficients were also completed for input into the CALFLOT program (see Attachment IV).

The data obtained for kinetic rate constants and the entrainment coefficients are now available for the CALFLOT program.

The program is called on the computer and its first request is for the type of monitor you are using (See Attachment VII). The "arrow" (cursor) keys are used, then the "esc" key instead of the normal "return" key for data input. It is important to note the "esc" key in this program is always used for input rather than the "return" key.

Method: Next, the program asks for the various options (shown in Attachment VII, page 1), and in this case, we use "Data Input". The program next requires the calculation "mode" (shown in Attachment VII, page 2), and then a selection of circuit configuration. The "Data Configuration" is next required. Two options are available; the "N" mode and the "Y" mode.

1. The "N" mode

   In "Data Configuration" in the "N" mode, we will not submit the kinetic rate data and the program uses internally generated numbers from previously carried kinetic rate test work which has been completed by Laval University.

   The program next requires "General Data (1)" (Attachment VII, page 3), which consists of feed rates and mineral information, and then description of densities and mineral feed grades as required in General Data (2).

   The program then asks for "Data" on each section of the circuit. A choice of minerals is given along with the percent of solids in the feed. This is completed for the rougher and all cleaners (Attachment VII, pages 4 and 5).

   The program completes the input section by asking for the "Restrictions and Control Parameters" (Attachment VII, page 6), which are set by Actual or Recommended Values. The computer will now return to the "Select Option" mode and you move to "Calculation".

2. The "Y" mode

   The program requires "Y's" in the Data Configuration section (as shown in Attachment VII, page 8). The computer next requests the "General Data (1) and (2)" the same as in the "N" mode, and then a "General Data (3)" for entrainment coefficients (Attachment VII, page 9).

   The computer then asks for "Data" on each section of the plant, requesting kinetic rate data and other information for each process circuit (as shown in Attachment VII, pages 10 and 11). The computer completes data input by asking for "Restrictions and Control Parameters" (as shown in Attachment VII, page 12) and then returns to the "Select Option" mode.

On return to the "Select Option" mode, we select "Calculation", which takes the data and runs it through the mathematical program. The program will take several iterations and many calculations for each data set, taking up to 5 minutes to complete the calculations (Attachment VIII).

Author(s) J. Taylor	*technical notes*	Project No.	Page of

Method:

When the calculations are complete the computer returns to the "Select Option" mode and we move to the "Print" mode (Attachment IX, page 1). The computer asks for the "Print Destination, Printing Content, Balance Format, and Graphic Content". The program will print out the "Input Data" (Attachment IX, pages 2 and 3), the "Equipment" (Attachment IX, page 4), and the "Balances" "Summary or Detailed" (as shown in Attachment IX, pages 5 to 10). Attachment X, pages 1 to 5, show examples of the "Flowsheet" and "Curves" available under the graphics display.

The CALFLOT system also has a "Select Option" for the "Saving, Reading, and Erasing" of files, where "Input Data" for each test can be stored for further processing at a later time.

An operating manual and program outputs for the CALFLOT computer program supplied by Laval University are given in Attachment XI.

ATTACHMENT I

SUBJECT: Mathematical-Graphical Method for Calculating Kinetic Rate Parameters

OBJECTIVE: To produce kinetic rate parameters from standard rate kinetic flotation tests using a mathematical-graphical method supplied by Laval University in conjunction with the CALFLOT program. The method will produce Kf (fast floating rate constant), Ks (slow floating rate constant), Ø (the amount of slow floating mineral), and R∞ (the maximum possible mineral recovery in the feed). This data can then be used by CALFLOT for flotation circuit design.

CONCLUSION: Through a series of mathematical and graphical solutions, we are able to obtain the required rate parameters.

DISCUSSION: The method of calculation shown here was supplied by Laval University in connection with our purchase of the CALFLOT program. The method has minor problems in obtaining accurate results due to interpolation of graphs used in the solution of the required constants. The method, however, agrees reasonably well with other methods of calculation.

METHOD: This mathematical-graphical method uses equations supplied by Mr. M. Bourassa of Laval University and is shown in Supplement I. The method used the equations shown in models 5, 6 and 7.

The first step in solving the rate kinetic problem is to determine the maximum recovery parameters-R∞. This is accomplished by plotting a graph of recovery at time "T" versus recovery at time T + Δt. In this case, we chose a Δt the same as the original test design flotation times. The data is recorded in the table below:

TIME IN MINUTES	RECOVERY AT TIME	RECOVERY AT TIME + Δt
4	0.519	0.6017
8	0.6017	0.6667
16	0.6667	0.7127
24	0.7127	0.7293

The data is then plotted, as shown in Figure 1. The intersection of the line of the data points and a line with a slope of 1 going through the origin will give an R_∞ value of approximately 77.4% or $R_\infty = 0.774$.

Once R_∞ has been determined, we use the model "6", as shown in Supplement I, and use the relationship

$$\ln(1 - \frac{R}{R_\infty}) = \ln Ø - Ks \cdot t$$

if R_∞ is known, and R then a plot of $\ln(1 - \frac{R}{R_\infty})$

versus time will give the intercept Ø, and -Ks as the slope of the line. The data is recorded in the table below:

TIME IN MINUTES	RECOVERY	$\ln(1 - \frac{R}{R_\infty})$
0.5	0.1852	not used
2.0	0.5190	not used
4.0	0.6017	not used
8.0	0.6667	-1.976
16.0	0.7127	-2.536
24.0	0.7293	-2.852

In this case, only the last three data points are used because we are looking for the slow floating material, and this will only occur at long flotation times.

The data was plotted on a graph, as shown in Figure 2, and from the graph, the Y-intercept is -1.65, therefore,

$$Ø = e^{-1.65} = 0.192$$

and the slope of the line is $\frac{-17.5 - (-2.875)}{2 - 24} = -0.051$

therefore, Ks = 0.051.

Now using the second part of model 6 in Supplement I, we know Ks and Ø, the equation is:

$$\ln\left[\frac{1-\frac{R}{R\infty}-\phi\left(1-e^{-kst}\right)}{1-\phi}\right]=-kft$$

By plotting on a graph:

$$\ln\left[\frac{1-\frac{R}{R\infty}-\phi\left(1-e^{-kst}\right)}{1-\phi}\right]$$

versus time, as shown in Figure 3, we can determine from the slope of this line -Kf

$$-Kf = \frac{-0.371 - (-4.41)}{0 - 8} = -0.505$$

Kf = 0.505

This completes the calculation of the rate constants, which are summarized below:

Kf = 0.505
Ks = 0.051
Ø = 0.192
R∞ = 0.774

SUPPLEMENT I

FACULTE DES SCIENCES ET DE GENIE

THESE
PRESENTEE
A L'ECOLE DES GRADUES
DE L'UNIVERSITE LAVAL
POUR L'OBTENTION
DU GRADE DE MAITRE ES SCIENCES (M.Sc.)

PAR

MICHEL BOURASSA
BACHELIER ES SCIENCES APPLIQUEES
DE L'UNIVERSITE LAVAL

ETUDE LABORATOIRE-USINE DE LA FLOTTATION DE LA CHALCOPYRITE A LAC MATTAGAMI

OCTOBRE 1986

Reprinted with authorization of the author:

Michel Bourassa, P. Eng.

ANNEXE 4

METHODES D'ESTIMATION DES PARAMETRES DES MODELES CINETIQUES

Pour chacun des modèles on présente une méthode simple pour estimer les paramètres. Dans certains cas il existe d'autres méthodes pour évaluer les paramètres mais celles décrites ont l'avantage d'être des méthodes graphiques facilement utilisables.

Modèle 1: $R=1-(\lambda/\lambda+t)^p$

En posant $Y=1-R$ et en prenant le logarithme on a:

$$\ln(Y) = p\cdot\ln(\lambda) - p\cdot\ln(\lambda+p)$$

Lorsque le temps est suffisamment grand $(t>4-5\lambda)$; $\ln(\lambda+t)\approx\ln(t)$. On a alors une relation linéaire en traçant Y en fonction de t sur papier logarithmique.

Relation linéaire : $\ln(Y) = p\cdot\ln(\lambda) - p\cdot\ln(t)$
ordonnée = $p\cdot\ln(\lambda)$
pente = $-p$

Pour améliorer la qualité de l'estimation, on répète la procédure en remplaçant t par $t+\lambda$ avec λ obtenu à l'itération précédente. La convergence est très rapide et deux itérations suffisent pour avoir de bons estimés. D'autres méthodes sont présenté par Inoue et Imaizumi[10] mais sont plus hasardeuses que celle-ci.

Modèle 2: $R=R_\infty[1-(\lambda/\lambda+t)^p]$

Les paramètres de ce modèle sont difficile à évaluer en raison de la présence du R_∞. Connaissant ce paramètre on trouve les autres paramètres de la même façon que pour le modèle 1. Pour évaluer R_∞ on peut soit utiliser le R_∞ d'un autre modèle ou y aller par tatonnement. On fixe alors R_∞ pour calculer λ et la méthode d'estimation est la même que le modèle 1 soit:

Relation lin/aire : $\ln(1-[R/R_\infty] = p\cdot\ln(\lambda) - p\cdot\ln(t)$

ordonnée = $p\cdot\ln(\lambda)$

pente = $-p$

Modèle 3: $R=R_\infty(1-[(1-\exp-k\cdot t)/k\cdot t])$

Lorsque le temps est suffisamment grand (>3-4 min.), le terme exponentiel devient négligeable et la relation obtenue est linéaire. En traçant le graphe de R en fonction de 1/t on trouve les paramètres en se servant des coefficients de la droite.

Relation linéaire: $R=R_\infty - R_\infty/k\cdot t$

ordonnée = R_∞

pente = $-R_\infty/k$

Modèle 4: $R=R_\infty\cdot t/k+t$

Ce modèle se linéarise facilement en traçant le graphe de 1/R en fonction de 1/t

Relation linéaire : $1/R = 1/R_\infty + k/R_\infty\cdot t$

ordonnée = $1/R_\infty$

pente = k/R_∞

Modèle 5 : $R=R_\infty(1-\exp-k\cdot t)$

en posant $t_i=t_{i-1}+\Delta$ avec Δ=constante

on a: $R_i=R_\infty[1 - \exp(-k\cdot t_{i-1})\cdot\exp(-k\cdot\Delta)]$

$$R_i=R_\infty[1 - \xi\cdot\exp-k\cdot t_{i-1}] \qquad (1)$$

avec $\xi=\exp(-k\cdot\Delta)$=constante

de plus, $R_{i-1}=R_\infty(1 - \exp(-k\cdot t_{i-1})$

$$\Longrightarrow \exp(-k\cdot t_{i-1}) = (R_\infty - R_{i-1})/R_\infty \qquad (2)$$

On trouve finalement, en remplaçant [1] dans [2], une relation linéaire entre R_t et R_{t-1}.

relation linéaire : $R_t=R_\infty(1-\xi) + \xi\cdot R_{t-1}$
ordonnée = $R_\infty(1-\xi)$
pente = ξ

Une fois qu'on connait R_∞, on trace sur une échelle semi-logarithmique $1-(R/R_\infty)$ en fonction du temps.

relation linéaire : $\ln[1 - (R/R_\infty)] = -k\cdot t$
ordonnée = 0
exp. pente = $-k$

Modèle 6: $R=(1-\phi)(1-\exp[-k_f\cdot t]) + \phi(1-\exp[-k_s\cdot t])$

Comme k_f est beaucoup plus grand que k_s (environ 10 fois), le terme $\exp(-k_f\cdot t)$ devient négligeable lorsque t est assez grand (t>3-4 min.). On obtient une relation linéaire en traçant 1-R en fonction de t sur une échelle semi- logarithmique.

Relation linéaire : $\ln(1-R)=\ln(\phi) - k_s\cdot t$ (t>3-4 min.)
ordonnée = $\ln(\phi)$
pente = $-k_s$

Lorsque l'on connait ϕ et k_s, on a une autre relation linéaire qui nous donne la valeur de k_f.

Relation linéaire : $\ln\{1- [R-\phi(1-\exp-k_s\cdot t)]/[1-\phi]\} = -k_f\cdot t$
③ ordonnée = 0
pente = $-k_f$

Modèle 7: $R=R_\infty\{(1-\phi)(1-\exp-k_f\cdot t)+\phi(1-\exp-k_s\cdot t)\}$

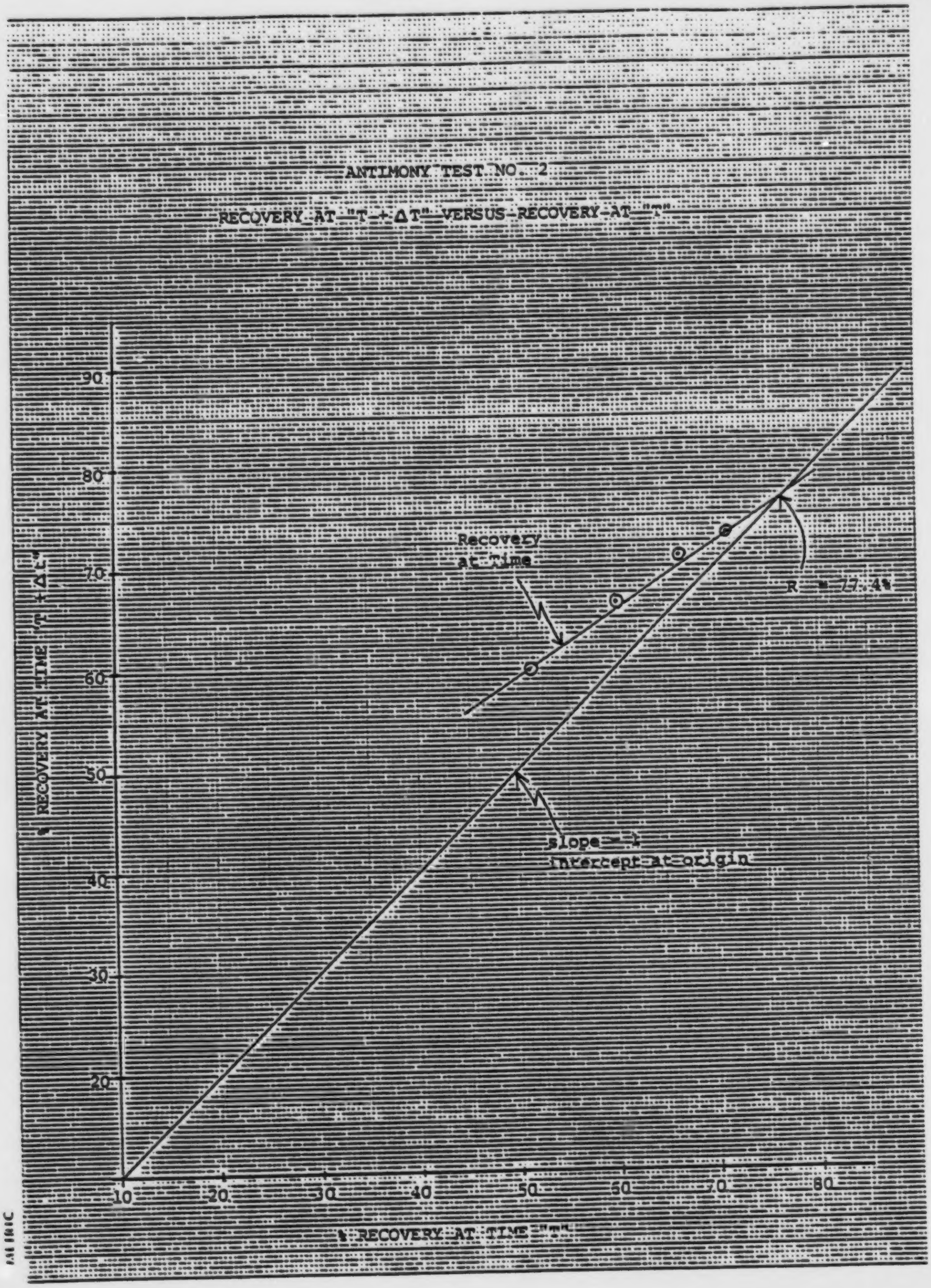
En faisant la même hypothèse que pour le modèle 6, lorsque t est grand le modèle devient:

$$R=R_\infty[1-\phi\cdot\exp(-k_s\cdot t)]$$

Le paramètre R_∞ se détermine de la même façon que pour celui du modèle 5. Une fois ce paramètre connu, le problème à résoudre est similaire à celui du modèle précédent (6).

$$R/R_\infty=(1-\phi)(1-\exp[-k_f\cdot t])+\phi(1-\exp[-k_s\cdot t])$$

FIGURE 1



ANTIMONY TEST NO. 2

RECOVERY AT "T + ΔT" VERSUS RECOVERY AT "T"

FIGURE 2

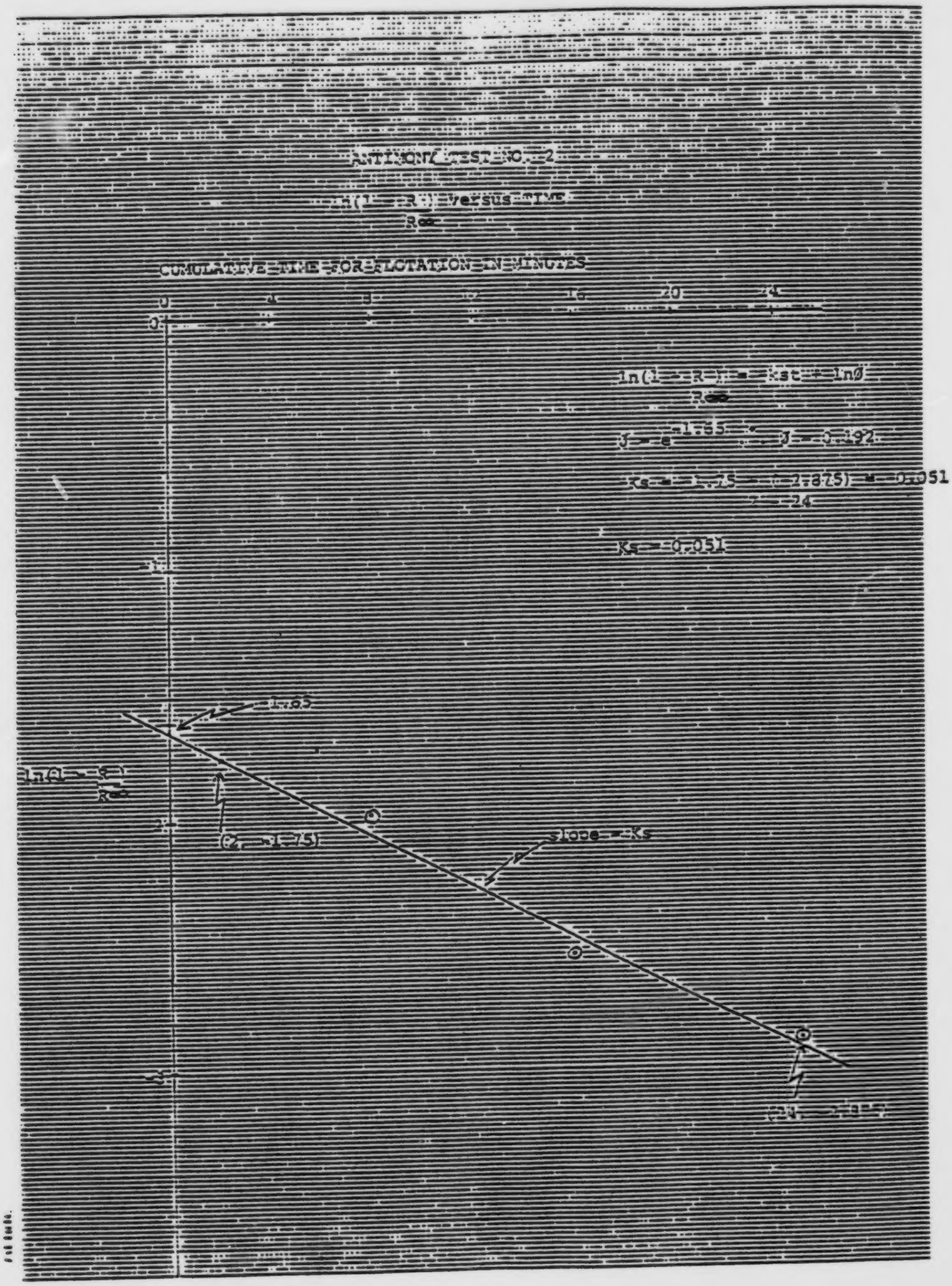
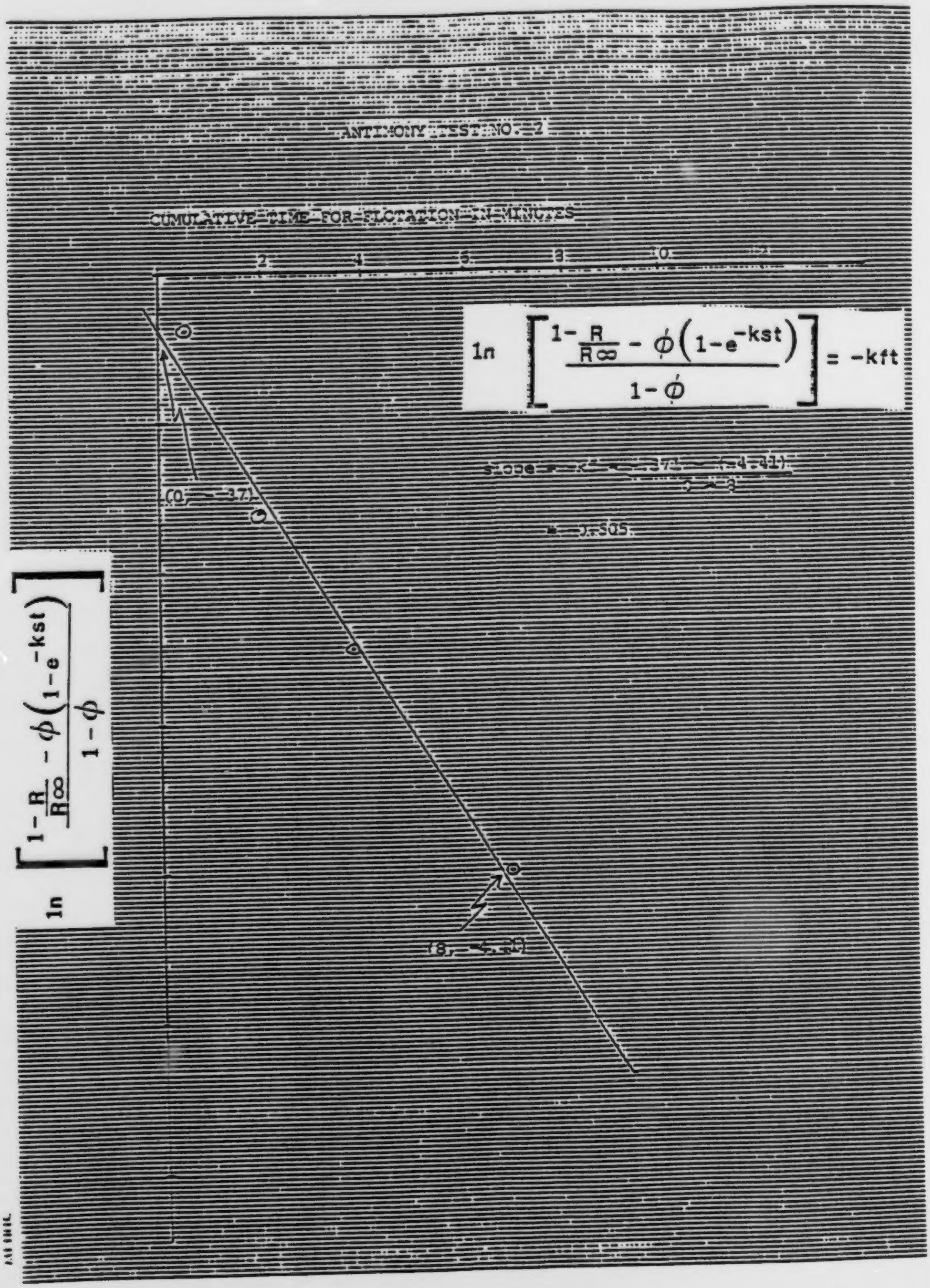

ANTIMONY TEST NO. 2
ln(1 - R/R∞) versus TIME
CUMULATIVE TIME FOR FLOTATION IN MINUTES
0
4
8
12
16
20
24
0
-1
-2
-3
ln(1 - R/R∞) = -Kst + lnØ
J = e^-1.65 ∴ J = 0.192
Ks = -1.75 - (-2.875) / 2 - 24 = -0.051
Ks = -0.051
-1.65
ln(1 - R/R∞)
(2, -1.75)
slope = Ks
(24, -2.875)

FIGURE 3


ANTIMONY TEST NO. 2
CUMULATIVE TIME FOR FLOTATION IN MINUTES
2
4
6
8
10
12
$\ln\left[\dfrac{1-\frac{R}{R\infty}-\phi\left(1-e^{-kst}\right)}{1-\phi}\right]=-kft$
slope = -kf = (-.37) - (-4.41) / 0 - 8
= -0.505
(0, -.37)
(8, -4.41)
$\ln\left[\dfrac{1-\frac{R}{R\infty}-\phi\left(1-e^{-kst}\right)}{1-\phi}\right]$

ATTACHMENT II

SUBJECT: Cell Concentration Method for Calculating Kinetic Rate Parameters

OBJECTIVE: To produce kinetic rate parameters from standard rate kinetic flotation tests, using the method of cell concentrations, as shown in Supplement II. The method will produce Kf (fast floating rate constant), Ks (slow floating rate constant), ϕ (the amount of slow floating mineral, and R^{∞} (the maximum possible mineral recovery in the feed). This data can then be used by CALFLOT for flotation circuit design.

CONCLUSION: Through the use of cell concentrations using a mathematical-graphical method, we are able to obtain the required rate parameters.

DISCUSSION: The method of calculation shown here was taken from the book "Mineral and Coal Flotation Circuits" by A.J. Lynch et. al published by Elsevier Scientific Publishing Company. This method was used to verify that the methods supplied by Laval University were reasonably accurate when calculating rate parameters. However, the method has some minor problems, the most sever being interpolation of graphs leading to errors in accurate results for the rate parameters. The method, however, agrees well with other methods of calculation.

METHOD: The mathematical-graphical method requires the initial concentration of the floatable species in the cell called "Co" to be calculated. In this case, we assume a maximum recovery of about 5% above the last test recovery, and in this case, the maximum recovery used is 77.4%. "Co" is then calculated on the basis of the free mineral species, in this case SbS, in which 71.7% of the weight is Sb.

As our assays are done on the basis of Sb, calculations are necessary to convert to the mineral species. For "Co" we take the feed assay (Figure 4) and calculate the weight of pure Sb = 3.46% Sb (head assay) x 77.4% (recovery maximum) x 990.0 g (weight of feed) = 26.58 g of Sb. This is now converted to SbS:

$$26.58 \times \frac{100}{71.7} = 37.06 \text{ g of Sbs.}$$

The amount of SbS removed at each test is calculated from the assay and weight of each concentrate sample taken during the test interval. At the time t = 0.5, the SbS is calculated by 10.48 g (weight of concentration) x 60.70% (assay of Sb) x $\frac{100}{71.7}$ (Sb to SbS) = 8.87 g SbS (see Figure 5 for data). This amount of SbS is subtracted from the original amount of SbS to give a new concentration in the cell of SbS. When all the data for each time interval is completed, then

the ratio of $\frac{C(t)}{Co} = \frac{\text{concentration at time (t)}}{\text{concentration at time (o)}}$ is found

The results are shown in Figure 5

When the ratio of $\frac{C(t)}{Co}$ has been calculated a graph of

$\frac{C(t)}{Co}$ is plotted versus time (Figure 6).

An estimate of Cs (concentration for slow floating mineral) is made by taking the last four data points and drawing a straight line through them to the Y-axis. From this line Cs the value of Cs is taken, as shown in Figure 6.

In Supplement I, the equation 3.16 is shown as
$C = Co \cdot [\phi \cdot \exp(-Ks \cdot t) + (1 - \phi) \cdot \exp(-Kf \cdot t)]$

in this equation: $Cs = \phi \cdot \exp(-Ks \cdot t)$
$Cf = (1 - \phi) \cdot \exp(-Kf \cdot t)$

Therefore, $\frac{C}{Co} = Cs + Cf$, and if we have

Cs from the graph and $\frac{C(t)}{Co}$, then Cf can

be found by subtraction and is shown in Figure 6. Note that only the first three data points are used, for that is only where the fast floating species would occur.

The rate kinetic terms can now be worked out.

(1) ϕ is where Cs crosses the Y-axis and in this case it is 0.225.

(2) The fast float rate constant Kf

$Cf = (1 - \emptyset) \cdot \exp(-Kf \cdot t)$, using $\emptyset = 0.225$ and the first three time intervals:

Time	Kf
t = 0.5	0.7244
t = 2.0	0.9560
t = 4.0	0.8270

Average = 0.836 = Kf

(3) The slow float rate constant Ks

$Cs = \emptyset \cdot \exp(-Ks \cdot t)$ using $\emptyset = 0.225$ and the last three time intervals:

Time	Kf
8.0	0.0402
16.0	0.0497
24.0	0.0858

Average = 0.0586 = Ks

(4) Data Summary

Kf = 0.836
Ks = 0.059
Ø = 0.229
R∞ = 0.774

The data is then run on CALFLOT to give a circuit design, as shown in Attachment VI.

SUPPLEMENT II

5.4 CALCULATION OF MODEL PARAMETERS

Procedures for the calculation of parameters in flotation models will be explained through numerical examples. There are many parameters which have been used in various models and only those which are regarded as important in circuit optimisation and control will be considered in the examples.

5.4.1 *Flotation rate constants*

The simple two rate constant model given in eq. 3.16 describes with reasonable accuracy the behaviour of the valuable particles in many semi-

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Title: Mineral and Coal Flotation Circuits - Their Simulation and Control

Series: Developments in Mineral Processing, Volume 3

Authors: A. J. Lynch, N.W. Johnson, E. V., Manlapig, C. G. Thorne

Pages: 125-128, 65-67

batch processes. For a continuous cell eq. 3.25 is the basis of the model

$$R_s = K_s \cdot \tau / (1 + K_s \cdot \tau)\ ;\ R_f = K_f \cdot \tau / (1 + K_f \cdot \tau)\ ;\ R = \phi \cdot R_s + (1 - \phi) \cdot R$$

The derivation of the parameters K_f, K_s and ϕ from plant data is given in Example 5-2.

Example 5-2: A sampling survey has been carried out on a rougher bank in which galena is floated and the following data are derived about bank performance from the completed mass balances.

Cell No.	1	2	3	4	5	6	7	8	9	10	11	12
C/C_0	0.556	0.312	0.201	0.135	0.098	0.083	0.069	0.055	0.051	0.044	0.039	0.035

The data are shown in Fig. 5-5. It may be assumed that the retention time is 20 seconds per cell. The problem is to calculate the parameters K_f, K_s and ϕ for the galena. The following sequence of calculations are carried out to solve the problem.

(1) K_s is estimated by assuming that only slow floating particles float in the last four cells. For these cells

$$R = (C_8 - C_{12}) / C_8 = 1 - (1 + K_s \cdot \tau)^{-N}$$

$$0.3636 = 1 - (1 + K_s \cdot 0.33)^{-4}$$

$$K_s = 0.362 \text{ min}^{(-1)} ✓$$

(2) The value of ϕ, which is the proportion of galena which has a low rate constant, is estimated either graphically by projecting the straight line connecting the concentrations at cells 8 and 12 on the concentration-cell number curve in Fig. 5-5 to intersect the ordinate, or algebraically as follows:

$$R = (\phi - C) / \phi = 1 - (1 + K_s \cdot \tau)^{-N}$$

For the slow floating fraction after 8 cells or 2.67 minutes retention time

$$(\phi - 0.055) / \phi = 1 - (1 + \underset{0.362}{\cancel{0.33}} \cdot \underset{0.33}{\tau})^{-8}$$

$$\phi = 0.1362 ✓$$

(3) The behaviour of the low rate constant fraction can be determined on a cell-by-cell basis. The equation

$$R_N = (\phi - C_N) / \phi = 1 - (1 + K_s \cdot \tau)^{-N}$$

The values at $C_0 - C_{12}$ are

(0.1362), 0.1218, 0.1086, 0.0970, 0.0867, 0.0773, 0.0680, 0.0617, 0.0550, 0.0491, 0.0439, 0.0392, 0.0350.

(4) The behaviour of the high rate constant fraction may be determined on cell-by-cell basis by subtracting the low rate constant fraction for the total particles. The value for the cell 1 tailing is 0.5560 − 0.1218 = 0.4362 and the values for each cell are

(0.8638), 0.4342, 0.2034, 0.1040, 0.0483, 0.0207, 0.0140, 0.0073.

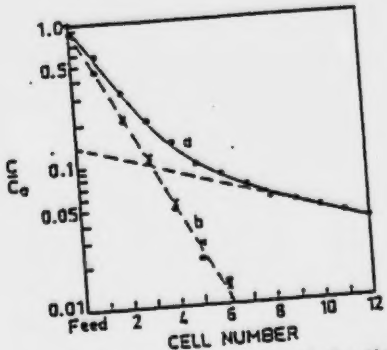


Fig. 5.5. Line a — observed fraction of galena in successive cells down a flotation bank.

Line b — observed and calculated behaviour of the fast-floating fraction after the behaviour of the slow-floating fraction is accounted for.

(5) K_f may now be calculated using the data from the first cell.

$$(0.8638 - 0.4342) / 0.8638 = 1 - [1 + 0.33 \cdot K_f]^{-1}$$

$K_f = 3.00\ \mathrm{min}^{-1}$

A comparison of the observed and calculated values of the high rate constant fraction, using the calculated value of K_f, as it proceeds down the bank is as follows.

Cell No.	1	2	3	4	5	6	7
Observed fraction in tailing	0.4342	0.2034	0.1040	0.0483	0.0207	0.0140	0.0073
Calculated fraction in tailing	0.4342	0.2151	0.1074	0.0541	0.0268	0.0134	0.0067

The agreement is shown in Fig. 5–5.

The best estimates of the model parameters K_f, K_s and ϕ may be calculated using a standard least squares fitting procedure. This involves minimissing $\Sigma_i\ (0_i - C_i)$ with respect to the model parameters. 0_i and C_i are the observed and calculated values at the i^{th} time interval. Deming (1964) may be consulted for the mathematical techniques.

The calculation of model parameters must be carried out with full recognition of the conditions of process operation as these may be such as to cause complications. For instance inhibited flotation conditions and/or froth overloading may occur early in the rougher and cleaner banks in sulphide flotation and this means that there is an additional mechanism operating which restricts the rates at which particles appear in the concentrates irrespective of their true flotation rates. Use of the normal procedure for the calculation of flotation rates for cells in which the additional mechanism is operating will lead to erroneous results. Another complication may occur when collector is added during the flotation process. This may affect any or all of the three parameters and it is necessary to recognise that different values of the parameters may exist before and after collector addition. Similarly variation in water flow to the concentrate caused by the addition of frother or by variation in pulp level during the process may cause difficulty in interpretation of data.

The procedures for including changes in these variables in the model are discussed in chapter 6.

The factors affecting the flotation rate constants are discussed in more detail in chapter 6. It is important to emphasise, however, that in some cases the assumption that valuable particles may be classified into two groups with high and low rate constants respectively may be invalid and in these cases one of the more complex models discussed in chapter 3 must be used for simulation purposes. For instance if the ore is very fine grained the occurrence of composite particles with a range of rate constants must be recognised. Numerical examples in this monograph will be restricted to the two rate constant case as this case covers most industrial operations which have been investigated.

particles are deposited into the concentrate launder at a rate which is proportional to the concentration of those particles in the pulp (pulp to froth transfer being the rate determing step).

The simple rate equation for a semi-batch flotation process may be written

$$-dC/dt = K \cdot C \tag{3.12}$$

where C is the concentration of particles with identical flotation properties in the pulp and t is time. The rate constant K is related to the flotation conditions such as the collector concentration and pulp level. The term $(-dC/dt)$ represents the instantaneous recovery rate from a semi-batch compartment.

If the term C_0 represents the initial concentration of particles with identical flotation properties in the pulp of a semi-batch compartment, the integrated form of the rate equation is

$$C = C_0 \cdot \exp(-K \cdot t) \tag{3.13}$$

The similarity between this equation (for a semi-batch compartment) and the probability equation proposed by Kelsall (for a bank of continuous compartments at steady state), eq. 3.11, will be noted in the expanded form of eq. 3.13 which is

$$C = C_0 \cdot [1 - K + \frac{K^2}{2!} - \frac{K^3}{3!} + \ldots]^t \tag{3.14}$$

and which approximates to $C = C_0 \cdot (1 - K)^t$ since terms in K^2 and above become small. The parameters C and W, t and n, and K and P are synonymous. Both equations assume plug flow characteristics for the compartment in their development (see also Section 3.4.2).

If the particles in the pulp do not possess identical flotation properties but possess a continuous range of rate constants, the integrated form of the rate equation for a semi-batch compartment becomes

$$C = C_0 \cdot \int_0^\infty \exp(-K \cdot t) f(K, 0)\, dk \tag{3.15}$$

where $f(K, 0)$ represents a continuous distribution of rate constants.

The major problem with eq. 3.15 is related to the accurate estimation of the distribution of the rate constants.

With regard to this distribution, Kelsall (1961) suggested that the approximation could frequently be made that the floatable species could be divided into two components with high and low flotation rate constants. In the cases where this approximation is valid the rate equation for a semi-batch laboratory cell may be written

$$C = C_0 \cdot [\phi \cdot \exp(-K_s \cdot t) + (1 - \phi) \cdot \exp(-K_f \cdot t)] \tag{3.16}$$

where K_s and K_f are the two rate constants and ϕ is the proportion of the floatable species with low rate constant K_s.

The concepts embodied in eqs. 3.13 and 3.16 are illustrated in the following numerical examples.

Example 3.1. The progress of semi-batch flotation under ideal conditions can be described by the first order rate equation

$$C = C_0 \cdot \exp(-K \cdot t)$$

where C and C_0 are the concentrations of the floatable component in the cell at time t and zero time respectively, and K is the flotation rate constant. Calculate and plot the mass fraction of the component in the cell after each minute of flotation up to 8 minutes when K is 0.5 and 1.0 min^{-1}.

Solution

The mass fraction C/C_0 of the species in the cell as a function of time for different flotation rate constants is as follows

t	1	2	3	4	5	6	7	8
$K = 1.0$ min^{-1}	0.368	0.135	0.049	0.018	0.007	0.002	0.0009	0.0003
$K = 0.5$ min^{-1}	0.606	0.368	0.223	0.135	0.082	0.049	0.030	0.018

The results are shown graphically in Fig. 3–4.

Example 3.2. In example 3.1 a single floatable component only was considered but in many real processes it has been found that the approximation may be made that the valuables in the feed are made up of two components which are fast and slow floating respectively. The equation describing the progress of flotation of this feed may be written

$$C = C_0 \cdot [\phi \cdot \exp(-K_s \cdot t) + (1 - \phi) \cdot \exp(-K_f \cdot t)]$$

where K_s and K_f are the flotation rate constants of the slow and fast floating components and ϕ is the fraction of the valuable species which is slow floating. Calculate and plot the mass fraction of the valuable component in the cell after each minute of flotation up to 8 minutes when K_f and K_s are 1.0 and 0.1 min^{-1} respectively and ϕ is 0.2.

Solution

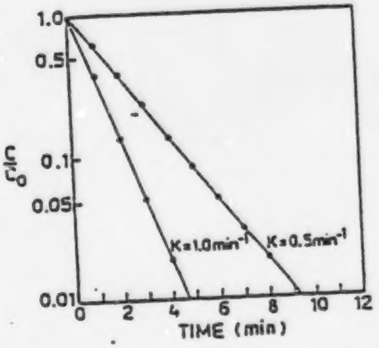
The mass fraction of the valuable component in the cell as a function of time may be calculated from the equation and the results are given in Table 3-I.

TABLE 3-I

Calculated concentration of the valuable component in the cell as a function of time when the component includes fast and slow floating species.

t (min)	$C_f = 0.8 \cdot e^{-1.0 \cdot t}$	$C_s = 0.2 \cdot e^{-0.1 \cdot t}$	$C/C_0 = C_f + C_s$
1	0.2943	0.1810	0.4753
2	0.1083	0.1637	0.2720
3	0.0398	0.1482	0.1880
4	0.0146	0.1341	0.1486
5	0.0054	0.1231	0.1267
6	0.0020	0.1098	0.1118
7	0.0007	0.0993	0.1000
8	0.0002	0.0899	0.0901

The results are shown graphically in Fig. 3-5.



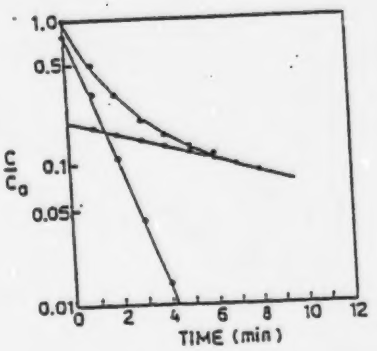


Fig. 3-4. Batch flotation with a single floatable species – effect of change in rate constant.

Fig. 3-5. Batch flotation with two floatable species having high and low rate constants.

3.4.2 *Residence time distribution in isolated continuous compartments*

The recovery of a mineral in a continuous compartment is governed not only by the distribution of rate constants describing particle floatabilities but also by the distribution of the retention times of particles in the pulp volume which depends on the type of mixing or flow regime (see Levenspiel, 1962, pp. 309–323). Thus for a single component with a distribution of residence times

$$C = C_0 \cdot \int_0^\infty \exp(-K \cdot t)\, E(t)\,dt \tag{3.17}$$

and for many components

$$C = C_0 \cdot \int_0^\infty \int_0^\infty \exp(-K \cdot t)\, E(t) f(K \cdot 0)\, dt\, dK \tag{3.18}$$

where terms C and C_0 can have units of concentration or flowrate. Therefore it is necessary to examine the type of mixing which exists in the pulp volume.

Danckwerts (1953) recognised two limiting conditions of mixing regimes in a continuous reactor. In one condition there exists a continuous gradation in the concentration of components over the length of reactor or isolated compartment. In such a reactor, all particles in the exit (tailing) stream have spent an equal time in the reactor (isolated compartment). In a semi-batch flotation compartment, all particles have also spent an equal time in the reactor. If all particles spend an equal time in a reactor, the reactor has piston, plug or tube flow mixing characteristics and the three terms are

FIGURE 4

GRADE AND RECOVERY

METAL: Sb   SAMPLE NAME ANIMONY FLOAT II   FILE NAME CALFLOT

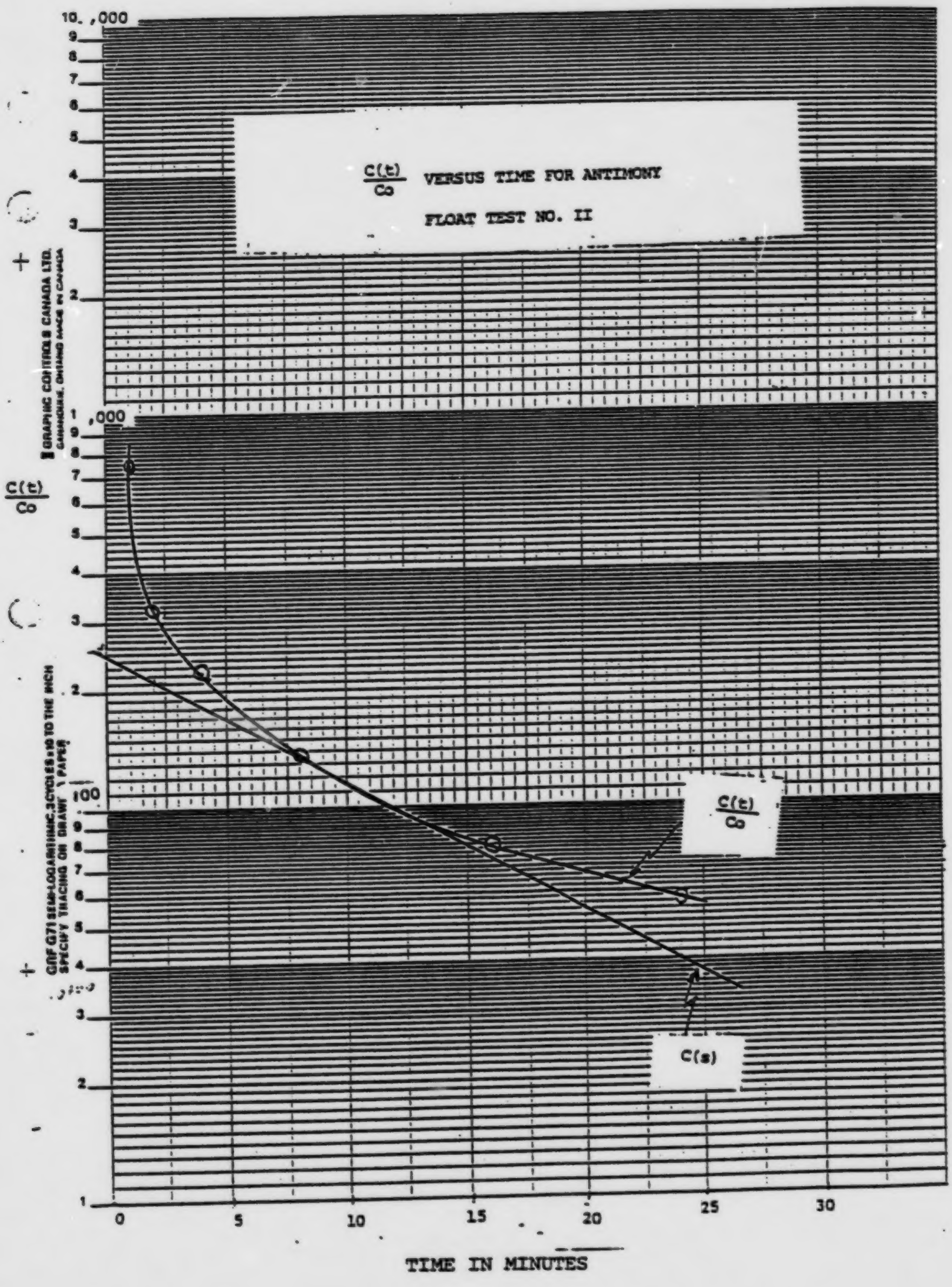
SAMPLE	WEIGHT (grams)	WEIGHT PERCENT	METAL ASSAY	METAL CONTENT	METAL DISTRIBUTION	GRADE	CUMULATIVE % WEIGHT	CUMULATIVE % RECOVERY
TAILS	929.82	93.92	1.00	93.92	27.07	41.62	6.08	72.93
24	5.45	0.55	10.40	5.73	1.65	0.00	5.53	71.27
.16	10.20	1.03	15.50	15.97	4.60	0.00	4.50	66.67
8	7.94	0.80	28.10	22.54	6.50	0.00	3.70	60.17
4	6.24	0.63	45.50	28.68	8.27	0.00	3.07	51.91
2	19.87	2.01	57.70	115.81	33.38	0.00	1.06	18.52
.30	10.48	1.06	60.70	64.25	18.52	0.00	-0.00	0.00
N/A	0.0C	0.00	0.00	0.00	0.00	0.00	-0.00	0.00
	0.00	0.00	0.00	0.00	0.00	0.00	-0.00	0.00
	0.00	0.00	0.00	0.00	0.00	0.00	-0.00	0.00
	0.00	0.00	0.00	0.00	0.00	0.00	-0.00	0.00
FEED	990.00		3.30	346.90				

BALANCE: 105.12

FIGURE 5

Cumulative Time in Minutes	Weight of SbS Removed in Concentrate	Weight of SbS Remaining in Flotation Cell	Ratio $\frac{C(t)}{C_o}$	Cs Concentration Slow Float	Cf Concentration Fast Float
0	0.0	37.06	1.00		
0.5	8.87	28.19	0.7606	0.2209	0.5397
2.0	15.99	12.20	0.3238	0.2093	0.1145
4.0	3.96	8.24	0.2223	0.1939	0.0284
8.0	3.11	5.13	0.1384	0.1631	
16.0	2.20	2.93	0.0791	0.1015	
24.0	0.790	2.14	0.0577	0.0399	

FIGURE 6



$\frac{C(t)}{Co}$ VERSUS TIME FOR ANTIMONY
FLOAT TEST NO. II
10,000
1,000
100
9
8
7
6
5
4
3
2
1
$\frac{C(t)}{Co}$
$\frac{C(t)}{Co}$
C(s)
0
5
10
15
20
25
30
TIME IN MINUTES
GRAPHIC CONTROLS CANADA LTD. GANANOQUE, ONTARIO MADE IN CANADA
GRF G71 SEMI-LOGARITHMIC, 3 CYCLES x 10 TO THE INCH SPECIFY TRACING OR DRAWING PAPER

```
Test Identification : ANTIMONY 2

  Cumulative Time            Cumulative Recovery
       (min)             measured         calculated
1      0.5               0.1852             .2025
2      2                 0.5191             .4989
3      4                 0.6017             .6142
4      8                 0.6667             .6673
5      16                0.7127             .7079
6      24                0.7293             .7313
7
8
9
10

Parameter       Start Value       Relative        Optimal Value
                                accuracy (%)
  Kf               1                5                 .795
  Ks               0.1              5                 .066
  Phi              0.2              5                 .214
 R inf.            0.75             5                 .765

        ROOT SQARE OF RESIDUES : 0.029879
```

```
Test Identification : ANTIMONY 2

  Cumulative Time            Cumulative Recovery
       (min)             measured         calculated
1      0.5               0.1852             .2034
2      2                 0.5191             .4991
3      4                 0.6017             .6136
4      8                 0.6667             .6674
5      16                0.7127             .7087
6      24                0.7293             .7311
7
8
9
10

Parameter       Start Value       Relative        Optimal Value
                                accuracy (%)
  Kf               1                1                 .806
  Ks               0.1              1                 .073
  Phi              0.2              1                 .215
 R inf.            0.75             1                 .759

        ROOT SQARE OF RESIDUES : 0.029864
```

ATTACHMENT IV

SUBJECT: Calculation of Water Entrainment Coefficients

OBJECTIVE: To calculate water entrainment coefficients from laboratory flotation tests that will be used in the CALFLOT flotation plant simulator.

CONCLUSION: Entrainment coefficients can be calculated using the method from "Mineral and Coal Flotation Circuits" by A.J. Lynch, *et*. *al*. (See Supplement III)

DISCUSSION: As part of the CALFLOT program, water entrainment coefficients are needed in the program input. The water entrainment coefficient is a measure of non-floating solids which are carried by the water component of the froth to the concentrate. This material (tails) dilutes the concentrate grade and increases the quantity of concentrate produced and is therefore important in flotation plant design.

METHOD: In the laboratory flotation tests, all solids and water used in the flotation test were weighed. This included the cell wash water and all cell make up water. The cell was maintained at a set level during the test by use of a level indicator in the cell. From the data on water, solids, weights, and assays, we were able to calculate the weight of water recovered with each concentrate sample and the water in the flotation cell. We were also able to obtain the amount of non-floating (tails) solids in both concentrates and flotation cell.

The entrainment coefficient is then calculated (Supplement I) as:

$$\frac{\dfrac{\text{weight of tails in concentrate}}{\text{weight of water in concentrate}}}{\dfrac{\text{weight of tails in float cell}}{\text{weight of water in float cell}}} = CF$$

The entrainment coefficient for each time interval was then calculated and then averaged for use in the CALFLOT program (See Figure 7).

SUPPLEMENT 3

5.4.3 *The classification function*

The entrainment of pulp in the froth, followed by drainage of some

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Title: Mineral and Coal Flotation Circuits - Their Simulation and Control

Series: Developments in Mineral Processing, Volume 3

Authors: A. J. Lynch, N.W. Johnson, E. V., Manlapig, C. G. Thorne

Pages: 129-132

TABLE 5–XI

Comparison of the observed and calculated values of $F(t)$, which is the fraction of tracer leaving the flotation bank at time t.

Time (sec)	Observed $F(t)$	Calculated $F(t)$
0	0.00	0.0104
30	0.013	0.0226
60	0.273	0.2937
90	0.677	0.6430
120	0.853	0.8569
150	0.924	0.9506
180	0.950	0.9846
210	0.965	0.9955
240	0.977	0.9988
300	0.990	0.9999
360	0.998	1.0000

particles back to the pulp and carry-over of the remaining entrained particles into the concentrate launder was discussed in section 3.4.4. For any process the mass of entrained particles carried into the concentrate is a function of the mass of water flowing into the concentrate and this was discussed quantitatively for the simple case of a single size fraction in example 3.5.

Entrained particles of different sizes drain to the pulp at different rates and eqs. 3.32 and 3.33 were developed to describe the behaviour in the flotation process of liberated, non-hydrophobic gangue particles. The equations are

$$\text{Rate of recovery of free gangue of size } i \text{ by entrainment} = RRW \cdot CF_i \cdot MFGP_i/MWATP$$

where
$$CF_i = \frac{(\text{Mass of free gangue per unit weight water})_{\text{Con}}}{(\text{Mass of free gangue per unit weight water})_{\text{Pulp}}}$$

The function CF_i is an important characteristic of an ore since it has a significant effect on the efficiency of mineral separation. The procedure for calculating CF_i is given in Example 5.4.

Example 5.4 – Calculation of CF_i: A lead-zinc ore flows through a bank of rougher flotation cells at 100 t/h in a pulp containing 37% by weight of solids. The assay of the feed is 6.3% Pb and 10.6 Zn, and the size distribution and assay of each size fraction in the feed are given in Table 5–XII.

The metal contents of the cell concentrate and tailing are

	Concentrate	*Tailing*
% Pb	77.5	2.0
% Zn	6.7	10.9

and the size distribution and assay of each size fraction in the concentrate are also given in Table 5–XII.

TABLE 5–XII

The size distributions of the rougher flotation bank feed and concentrate, and the assays of each size fraction

Size (μm)	+105	105/75	75/53	53/43	43/35	35/27	27/20	20/16	−16
Feed									
% wt	43.7	12.6	8.8	6.6	6.0	5.8	4.7	2.6	9.0
% *Pb*	1.2	4.8	8.1	18.6	10.4	8.8	8.4	9.7	13.8
% *Zn*	8.2	13.0	12.6	16.0	12.6	12.8	14.1	13.3	12.4
Concentrate									
% wt.	7.7	13.0	14.0	20.7	17.3	7.3	5.8	3.0	11.2
% *Pb*	70.0	81.2	82.5	84.7	82.0	74.6	69.0	63.7	62.0
% *Zn*	8.9	3.2	2.3	1.0	2.9	4.2	5.4	6.8	7.3

The concentrate contains 66.4% solids by weight. Calculate the values in the function CF_i if the lead mineral, galena, contains 86.6% by weight of lead and the zinc mineral, marmatite, contains 56.5% by weight of zinc. It may be assumed that all non-sulphide gangue is in liberated form. The error implicit in this assumption will be discussed at the end of the calculation.

Step 1

The fraction and mass flow rate of non-sulphide gangue in each size range in the feed may be calculated from the *Pb* and *Zn* assays of each size range, the sizing distribution and the mass flow rate of feed. The values are given in Table 5–XIII.

TABLE 5–XIII

The mass flow rates of non-sulphide gangue in each size fraction in the flotation feed

Size (μm)	+105	105/75	75/53	53/43	43/35	35/27	27/20	20/16	−16
% Wt.	43.7	12.6	8.8	6.6	6.0	5.8	4.7	2.6	9.0
Frac. gangue	.840	.715	.683	.501	.657	.672	.653	.653	.622
T/h gangue	36.71	9.01	6.01	3.31	3.94	3.90	3.07	1.70	5.60

Step 2

The mass flow rates of solids and water in the concentrate, and the mass fraction of water entering the concentrate may be calculated

t/h solids in concentrate (using *Pb* assays) = $100 \cdot (6.3 - 2.0)/(77.5 - 2.0) = 5.70$ t/h

t/h water in concentrate = $5.7 \cdot 33.6/66.4 = 2.88$ t/h

mass fraction of water in feed which enters the concentrate = 0.0288

Step 3

The mass flow rate of non-sulphide gangue in each size fraction in the concentrate may now be calculated from the total mass flow rate and sizing distribution of the concentrate, and the assays of each size fraction. Results are given in Table 5–XIV.

TABLE 5–XIV

Mass flow rate of non-sulphide gangue in each size fraction in the concentrate.

Size (μm)	+105	105/75	75/53	53/43	43/35	35/27	27/20	20/16	−16
% Wt.	7.7	13.0	14.0	20.7	17.3	7.3	5.8	3.0	11.2
t/h–total	.440	.744	.801	1.184	.990	.418	.332	.172	.641
t/h–galena ($\cdot 10^{-4}$)	3558	6977	7632	11574	9379	3599	2645	1265	4587
t/h–marmatite ($\cdot 10^{-4}$)	693	421	326	210	491	311	317	207	828
t/h–n.-s. gangue ($\cdot 10^{-4}$)	149	41.8	52.3	55.6	130	270	358	248	995

Step 4

CF_i may be calculated from the following equation and the results are given below.

$$CF_i = \frac{(\text{t/h gangue in size fraction } i \text{ in concentrate})}{(\text{t/h gangue in size fraction } i \text{ in feed}) \cdot (\text{Frac. recovery water in concentrate})}$$

The assumption that gangue in tailing and feed are equal involves little error.

Size (μm)	+105	105/75	75/53	53/43	43/35	35/27	27/20	20/16	−16
CF_i	.0141	.0161	.0302	.0583	.1146	.2404	.4049	.5065	.6169

These values are plotted in Fig. 5–6. The calculated value of CF_i may now be used to calculate the behaviour of the non-sulphide gangue in the flow rate of water to the concentrate changes.

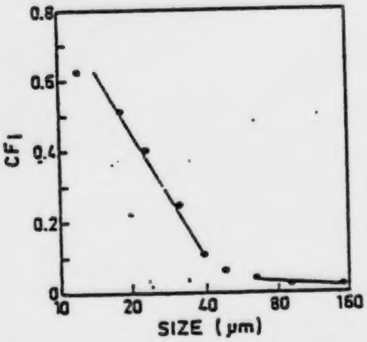


Fig. 5–6. The classification function calculated from plant data.

It will be noted that the CF_i-size relationship consists of two straight lines, one with a steep slope and the other with a shallow slope. The former refers to gangue particles which are liberated and appear in the concentrate by entrainment and classification, the latter to gangue which are in composite particles with sulphides and to which bubble-mineral aggregate occurs. In the case of the ore considered in this calculation the assumption that gangue particles greater than 50 μm are predominantly liberated appears to be incorrect.

FIGURE 7

ANTIMONY II

CONCENTRATE						TAIL				
TIME	WT SOLIDS	ASSAY	WT SbS	WT SiO_2	WT H_2O	WT SOLIDS	WT SbS REMAIN IN CELL	WT SiO_2	WT H_2O	CF
0.5	10.48	60.70	8.87	1.61	35.72	980.82	39.01	941.81	4274.14	0.2045
2.0	19.87	57.70	15.99	3.88	110.62	960.95	23.02	937.93	4193.84	0.1568
4.0	6.24	45.50	3.96	2.28	82.25	954.71	19.06	935.65	4170.07	0.1235
8.0	7.94	28.10	3.11	4.83	152.21	946.77	15.95	930.82	4138.26	0.1411
16.0	10.20	15.50	2.20	8.00	299.30	936.57	13.75	922.82	4170.83	0.1208
24.0	5.45	10.40	0.79	4.66	204.28	931.12	12.96	918.16	4127.29	0.1025
TAIL TOTAL	929.82 990.00	1.00	12.96 47.88	916.86 942.12						x=0.1415 s=0.03599

$$\text{wt solids} \times \text{assay} \times \frac{100}{71.7} = \text{wt SbS}$$

ATTACHMENT V

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE:	Antimony Feed	**PROJECT NUMBER:**	
SAMPLE WEIGHT:	1,000 g	**TEST NUMBER:**	Antimony II
PARTICLE SIZE:	1 hr grind	**TEST BY:**	B. Blizzard
PURPOSE:	Rate Parameter Tests on Antimony Ore	**DATE:**	May 26, 1988
BACKGROUND:		**CLIENT:**	Dominion Explorers

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)						
					Aero Drum 3477 10%	MIBC	Pb Nitrate				
Reagents			Natural		0.13 g	0.03 g	0.62 g				
Float 1	0.50										
Float 2	1.5										
Float 3	2.0										
Float 4	4.0										
Reagents					0.13 g	0.015 g	0.31 g				
Float 5	6.0										
Float 6	8.0										

COMMENTS: Reagent dosage split one-half at start one-half after 8 minutes of total flotation time.

ATTACHMENT VI

GRADE AND RECOVERY

METAL: Sb SAMPLE NAME ANIMONY FLOAT II FILE NAME CALFLOT

SAMPLE	WEIGHT (grams)	WEIGHT PERCENT	METAL ASSAY	METAL CONTENT	METAL DISTRIBUTION	GRADE	CUMULATIVE % WEIGHT	CUMULATIVE % RECOVERY
TAILS	929.82	93.92	1.00	93.92	27.07	41.62	6.08	72.93
24	5.45	0.55	10.40	5.73	1.65	0.00	5.53	71.27
16	10.20	1.03	15.50	15.97	4.60	0.00	4.50	66.67
8	7.94	0.80	28.10	22.54	6.50	0.00	3.70	60.17
4	6.24	0.63	45.50	28.68	8.27	0.00	3.07	51.91
2	19.87	2.01	57.70	115.81	33.38	0.00	1.06	18.52
.30	10.48	1.06	60.70	64.25	18.52	0.00	-0.00	0.00
N/A	0.00	0.00	0.00	0.00	0.00	0.00	-0.00	0.00
	0.00	0.00	0.00	0.00	0.00	0.00	-0.00	0.00
	0.00	0.00	0.00	0.00	0.00	0.00	-0.00	0.00
	0.00	0.00	0.00	0.00	0.00	0.00	-0.00	0.00
FEED	990.00		3.30	346.90				

BALANCE: 105.12

ATTACHMENT VII

```
                    ***********
                    *         *
                    * CALFLOT *
                    *         *
                    ***********

        SOFTWARE FOR CALCULATION OF FLOTATION CIRCUITS

                      (Release 2.0)
                  Serial Number:MB028870

   MICHEL BOURASSA, CRM, Energie et Ressources, Qubec, Canada
    GILLES BARBERY, GRAIIM, Universit Laval, Qubec, Canada

               Select the Monitor then <ESC>

                     COLOR CGA
                 ==> MONOCHROME
                     COLOR EGA

               FOR HELP ANYTIME : CTRL-H
```

```
               SELECT ONE OPTION THEN ESC


                     DATA INPUT
                     CALCULATION
                     PRINT
                     READ FILE
                     SAVE FILE
                     ERASE FILE
                     EXIT
```

```
                 CALCULATION MODE and CIRCUIT CONFIGURATION

=====>  SELECT CALCULATION MODE

                             Rougher-Scavenger Alone
                         ==> Rougher-Scavenger and Cleaner(s)

=====>  SELECT CIRCUIT CONFIGURATION

                             With Scavenger
ROUGHER                      No Scavenger
                         ==> Chose by Programm

                             With Scavenger, Conc. Return to Rougher Fee
1st CLEANER                  With Scav., Conc. Return to Cleaner 1 Feed
                             No Scavenger, Tail Return to Rougher Feed
                             No Scavenger, Tail Discarded
                         ==> Chose by Programm

2nd and 3rd CLEANERS         With 2nd Cleaner Alone
                             With 2nd and 3rd Cleaners
                         ==> Chose by Programm



                          DATA CONFIGURATIONS


                                                          RGHR  CLR1  CLR2  CLR2
Kinetic Parametres Known (R.inf.,Kf,Ks,Phi) ?              N     N     N     N
Entrainment Coef. for Unfloat minerals Known (CF) ?        N     -     -     -
Rate Constants for Water known ?                           N     N     N     N
Volume Cells fixed ?                                       N     N     N     N
Mark of Cells fixed ?                                      N     N     N     N
Number of Cells per Bank fixed ?                           N     N     N     N
Number of Cells Fixed ?                                    N     N     N     N
Water Flowrate for Froth Collapse known ?                  N     N     N     N
Proportion of Slow Floating in mids known ?                N     N     -     -
Number of Cells for Scavenger fixed ?                      N     N     -     -
Minimum Grade for Concentrate fixed ?                      N     N     N     N

                          - : Not applicable
```

GENERAL DATA (1)

Solid Flowrate (T.M./Hre)	100
Number of Interval Size Distributions (Max 5)	1
Number of Float Minerals (Max 2)	1
Number of Unfloat Minerals (Max 5)	2

Name of Float Mineral 1	CHALCOPYR
Name of Unfloat Mineral 1	CALCITE
Name of Unfloat Mineral 2	PYRITE

Minimum Grade of CHALCOPYR in the Final concentrate (frac.) ? 0.85

GENERAL DATA (2)

RELATIVE DENSITY

CHALCOPYR	4.2
CALCITE	2.7
PYRITE	5

MINERAL GRADES IN FEED (Fraction)

CHALCOPYR	.2000
CALCITE	.6500
PYRITE	.1500

	1.00

DATA FOR ROUGHER

CHALCOPY

Select the mineral who
is similar than yours ==>

Reference Minerals
Barytine
Coal
Chalcopyrite
Chalcopyrite Slow Floating
Fluorite
Nickel
Galena
Galena Slow Floating
Sphalrite (blende)
Wolframite-scheelite

Percent of Solids in Feed (f.) .3500

DATA FOR CLEANER 1

CHALCOPY

Select the mineral who
is similar than yours ==>

Reference Minerals
Barytine
Coal
Chalcopyrite
Chalcopyrite Slow Floating
Fluorite
Nickel
Galena
Galena Slow Floating
Sphalrite (blende)
Wolframite-scheelite

Percent of Solids in Feed (f.) .3000

DATA FOR CLEANER 2

CHALCOPY

Select the mineral who
is similar than yours

==>

Reference Minerals
Barytine
Coal
Chalcopyrite
Chalcopyrite Slow Floating
Fluorite
Nickel
Galena
Galena Slow Floating
Sphalrite (blende)
Wolframite-scheelite

Percent of Solids in Feed (f.) .3000

DATA FOR CLEANER 3

CHALCOPY

Select the mineral who
is similar than yours

==>

Reference Minerals
Barytine
Coal
Chalcopyrite
Chalcopyrite Slow Floating
Fluorite
Nickel
Galena
Galena Slow Floating
Sphalrite (blende)
Wolframite-scheelite

Percent of Solids in Feed (f.) .2800

```
                      RESTRICTIONS AND CONTROL PARAMETRES

                                 RESTRICTIONS

         SELECT DESIRED VALUES              Actual Values
                                       ==>  Recommended Values

                                               EBAUCH    NET1     NET2     NET3
Maximum Recovery for Cells (f)                  .3000    .4000    .4000    .4000
Minimum Recovery Justified for a Cell (f)       .0050    .0050    .0050    .0050
Max Percent Solid for Concentrates (f)          .3500    .4000    .4000    .4000

                              CONTROL PARAMETRES

         SELECT DESIRED VALUES              Actual Values
                                       ==>  Recommanded Values
     Tolerance for Residence Time Calcul. (f)            .0100
     Tolerance for Entrainment Calculation (f)           .0100
     Tolerance for Circulating Flow Calcul. (f)          .0100
     Max Iterations for Residence Time Calcul.               1
     Max Iterations for Entrainment Calculation              8
     Max Iterations for Circulating Flow Calcul.            10
```

```
                  CALCULATION MODE and CIRCUIT CONFIGURATION

=====>  SELECT CALCULATION MODE

                               Rougher-Scavenger Alone
                           ==> Rougher-Scavenger and Cleaner(s)

=====>  SELECT CIRCUIT CONFIGURATION

ROUGHER                        With Scavenger
                               No Scavenger
                           ==> Chose by Programm

1st CLEANER                    With Scavenger, Conc. Return to Rougher Fee
                               With Scav., Conc. Return to Cleaner 1 Feed
                               No Scavenger, Tail Return to Rougher Feed
                               No Scavenger, Tail Discarded
                           ==> Chose by Programm

2nd and 3rd CLEANERS           With 2nd Cleaner Alone
                               With 2nd and 3rd Cleaners
                           ==> Chose by Programm



                          DATA CONFIGURATIONS

                                                        RGHR  CLR1  CLR2  CLR2
Kinetic Parametres Known (R.inf.,Kf,Ks,Phi) ?            Y     Y     Y     Y
Entrainment Coef. for Unfloat minerals Known (CF) ?      Y     -     -     -
Rate Constants for Water known ?                         Y     Y     Y     Y
Volume Cells fixed ?                                     Y     Y     Y     Y
Mark of Cells fixed ?                                    Y     Y     Y     Y
Number of Cells per Bank fixed ?                         Y     Y     Y     Y
Number of Cells Fixed ?                                  Y     Y     Y     Y
Water Flowrate for Froth Collapse known ?                Y     Y     Y     Y
Proportion of Slow Floating in mids known ?              Y     Y     -     -
Number of Cells for Scavenger fixed ?                    Y     Y     -     -
Minimum Grade for Concentrate fixed ?                    Y     Y     Y     Y

                          - : Not applicable
```

```
                              GENERAL DATA (1)

      Solid Flowrate (T.M./Hre)                                  100
      Number of Interval Size Distributions (Max 5)              1
      Number of Float Minerals (Max 2)                           1
      Number of Unfloat Minerals (Max 5)                         2

                 Name of Float Mineral 1                    CHALCOPYR

                 Name of Unfloat Mineral 1                  CALCITE
                 Name of Unfloat Mineral 2                  PYRITE

    Minimum Grade of CHALCOPYR in the Final concentrate (frac.) ?  0.85



                              GENERAL DATA (2)

              RELATIVE DENSITY

             CHALCOPYR    4.2
             CALCITE      2.7
             PYRITE       5

   MINERAL GRADES IN FEED (Fraction)

        CHALCOPYR           .2000
        CALCITE             .6500
        PYRITE              .1500
                           ------
                            1.00



                              GENERAL DATA (3)

                    Entraiment coefficients for Unfloat Minerals


                              CALCITE       .70
                              PYRITE        .70
```

```
                         DATA FOR ROUGHER


                 CHALCO

Kf:Fast Rate(l/min)    .729
Scale-up Fact on Kf   1.000
Ks:Slow Rate(l/min)    .073
Scale-up Fact on Ks   1.000
Slow prop. (f)         .200
Slow Prop in mids      .200
Ultimate Rec. (f)      .950

                                                  .3500
Percent of Solids in Feed (f.)                    .0000
Water Flowrate to Collapse Froth (T.M./h)         .0050     Select the Mark
Water rate constant (l/min.)                     1.0000     of Cells
Volume of Cells (c. metre)                        .7700
Prop. of True Volume (f)                         3.0000   ==>  WEMCO
Number of cells par Bank                         1.0000        AGITAIR
Nb of Paral. Banks                                .0000        OUTOKUMPU
Nb of Scavenger Cells                             .0000        DORR
Minimum Grade of CHALCOPYR                        .0000
Minimum Grade of GALENE


                         DATA FOR CLEANER 1


                 CHALCO

Kf:Fast Rate(l/min)   1.184
Scale-up Fact on Kf   1.000
Ks:Slow Rate(l/min)    .118
Scale-up Fact on Ks   1.000
Slow prop. (f)         .200
Slow Prop in mids      .200
Ultimate Rec. (f)     1.000

                                                  .3000
Percent of Solids in Feed (f.)                    .0000
Water Flowrate to Collapse Froth (T.M./h)         .0080     Select the Mark
Water rate constant (l/min.)                     3.0000     of Cells
Volume of Cells (c. metre)                        .7500
Prop. of True Volume (f)                         4.0000        WEMCO
Number of cells par Bank                          .0000   ==>  AGITAIR
Nb of Paral. Banks                               1.0000        OUTOKUMPU
Nb of Scavenger Cells                             .0000        DORR
Minimum Grade of CHALCOPYR                        .0000
Minimum Grade of GALENE
```

DATA FOR CLEANER 2

```
                          CHALCO

Kf:Fast Rate(l/min)       1.184
Scale-up Fact on Kf       1.000
Ks:Slow Rate(l/min)        .118
Scale-up Fact on Ks       1.000
Slow prop. (f)             .200
Ultimate Rec. (f)         1.000


Percent of Solids in Feed (f.)                     .3000
Water Flowrate to Collapse Froth (T.M./h)          .0000
Water rate constant (l/min.)                       .0100       Select the Mark
Volume of Cells (c. metre)                        3.0000       of Cells
Prop. of True Volume (f)                           .7500
Number of cells par Bank                          5.0000           WEMCO
Nb of Paral. Banks                                 .0000           AGITAIR
Minimum Grade of CHALCOPYR                         .0000       ==> OUTOKUMPU
Minimum Grade of GALENE                            .0000           DORR
```

DATA FOR CLEANER 3

```
                          CHALCO

Kf:Fast Rate(l/min)       1.184
Scale-up Fact on Kf       1.000
Ks:Slow Rate(l/min)        .118
Scale-up Fact on Ks       1.000
Slow prop. (f)             .200
Ultimate Rec. (f)         1.000


Percent of Solids in Feed (f.)                     .2800
Water Flowrate to Collapse Froth (T.M./h)          .0000
Water rate constant (l/min.)                       .0100       Select the Mark
Volume of Cells (c. metre)                        1.0000       of Cells
Prop. of True Volume (f)                           .7500
Number of cells par Bank                          6.0000           WEMCO
Nb of Paral. Banks                                2.0000           AGITAIR
Minimum Grade of CHALCOPYR                         .0000       ==> OUTOKUMPU
Minimum Grade of GALENE                            .0000           DORR
```

```
                    RESTRICTIONS AND CONTROL PARAMETRES

                              RESTRICTIONS

          SELECT DESIRED VALUES              Actual Values
                                        ==>  Recommended Values

                                                EBAUCH   NET1    NET2    NET3
Maximum Recovery for Cells (f)                   .3000   .4000   .4000   .4000
Minimum Recovery Justified for a Cell (f)        .0050   .0050   .0050   .0050
Max Percent Solid for Concentrates (f)           .3500   .4000   .4000   .4000

                         CONTROL PARAMETRES

          SELECT DESIRED VALUES              Actual Values
                                        ==>  Recommanded Values
      Tolerance for Residence Time Calcul. (f)          .0100
      Tolerance for Entrainment Calculation (f)         .0100
      Tolerance for Circulating Flow Calcul. (f)        .0100
      Max Iterations for Residence Time Calcul.             1
      Max Iterations for Entrainment Calculation            8
      Max Iterations for Circulating Flow Calcul.          10
```

ATTACHMENT VIII

```
SELECT ONE OPTION THEN ESC


          DATA INPUT
   ==>    CALCULATION
          PRINT
          READ FILE
          SAVE FILE
          ERASE FILE
          EXIT
```

```
        ITERATION   3

Rougher-Scavenger       11

Cleaner 1-Scavenger      9

Cleaner 2                7

Cleaner 3                6
```

ATTACHMENT IX

```
                    SELECT ONE OPTION THEN ESC


                          DATA INPUT
                          CALCULATION
                   ==>    PRINT
                          READ FILE
                          SAVE FILE
                          ERASE FILE
                          EXIT
```

```
                    ******   PRINT   ******

                                   Select Flowsheet Informations
PRINT DESTINATION                     Flow       ==>CHALCOPYR
     Screen                       ==> Dist           SPHALERIT
 ==> Printer                          Grade          PYRITE
                                                     SILICATES
PRINTING CONTENT                                     WATER
     Input Data                                     SOLID
     Rsults: Equipment
     Rsults: Balance
 ==> All

BALANCE FORMAT
     Simplified Table
     Detailed Table
 ==> Graphics (with Specified Graphic Card)

GRAPHIC CONTENT
 ==> Flowsheet with Flowrate, Grade or Dist.
     Curve of GRADE-DISTRIBUTION vs CELL
```

```
                               -2-
                  ****** PRINT : INPUT DATA ******


   CALCULATION MODE: Rougher-Scavenger and Cleaner

   CIRCUIT CONFIGURATION
ROUGHER: Chose by Programm
1st CLEANER:  Chose by Programm
2nd and 3rd Cleaners:  Chose by Programm
                                                              RS  C1  C2
   DATA CONFIGURATION:
 Kinetic Parametres Known (R.inf.,Kf,Ks,Phi) ?                 N   N   N
 Entrainment Coef. for Unfloat minerals Known (CF) ?           N   -   -
 Rate Constants for Water known ?                              N   N   N
 Volume Cells fixed ?                                          N   N   N
 Mark of Cells fixed ?                                         N   N   N
 Number of Cells per Bank fixed ?                              N   N   N
 Number of Cells Fixed ?                                       N   N   N
 Water Flowrate for Froth Collapse known ?                     N   N   N
 Proportion of Slow Floating in mids known ?                   N   N   -
 Number of Cells for Scavenger fixed ?                         N   N   -
 Minimum Grade for Concentrate fixed ?                         N   N   N



 Solid Flowrate (T.M./h)=                    150
 Min. Grade of CHALCOPYR in the Final Conc (%)=  85

          Relative Density
         CHALCOPYR    4.2
         SPHALERIT    4
         PYRITE       5
         SILICATES    2.7

                          MINERAL FEED GRADE (frac.)

     CHALCOPYR          .0500
     SPHALERIT          .0500
     PYRITE             .2000
     SILICATES          .7000
                        ------
                        1.000


                          Entrainment Coefficients (CF)


                             SPHALERI     .70
                             PYRITE       .70
                             SILICATE     .90
                          RGHR      CLR 1     CLR 2     CLR 3
                          CHALCO    CHALCO    CHALCO    CHALCO

Kf:Fast Rate(l/min)        .729     1.184     1.184     1.184
Scale-up Fact on Kf       1.000     1.000     1.000     1.000
Ks:Slow Rate(l/min)        .073      .118      .118      .118
Scale-up Fact on Ks       1.000     1.000     1.000     1.000
Slow prop. (f)             .200      .200      .200      .200
Slow Prop in mids          .200      .200      .200      .200
Ultimate Rec. (f)          .950     1.000     1.000     1.000
```

```
                                          RGHR     CLNR1     CLNR2     CLNR3
Percent of Solids in Feed (f.)            .420      .300      .300      .280
Water rate constant (l/min.)              .005      .008      .010      .010
```

```
                                                 RGHR     CLNR1     CLNR2     CLNR3
Maximum Recovery for Cells (f)                   .300      .400      .400      .400
Minimum Recovery Justified for a Cell (f)       .0050     .0050     .0050     .0050
Max Percent Solid for Concentrates (f)           .350      .400      .400      .400
Tolerance for Residence Time Calcul. (f)        .0100
Tolerance for Entrainment Calculation (f)       .0100
Tolerance for Circulating Flow Calcul. (f)      .0100
Max Iterations for Residence Time Calcul.           1
Max Iterations for Entrainment Calculation          8
Max Iterations for Circulating Flow Calcul.        10
```

```
                           ****** PRINT : EQUIPEMENT ******

       CIRCUIT                        Rghr   Scav   Clnr1   Scav   Clnr2   Clnr3
Number of Paral. Banks:                1      1      1       1      1       1
Number of Cells/Bank  :                4      7      4       3      6       6
Volume of Cells (m3)  :              14.16  14.16   1.73    1.73    .64     .64
                (pi3) :              500.0  500.0   61.0    61.0   22.5    22.5
Type of Cells:                      simple simple simple  simple double  double
Power Installed (HP):                40.0   40.0   10.0    10.0    7.5     7.5
Manufacturer(s) * :                   WA     WA     W       W       A       A
Flotation Time (min):                 9.3   18.1    5.8     5.3    8.1    11.3
Unit Flotation Time :                2.32   2.58   1.46    1.75   1.36    1.88
Alternative Volume(m3):              7.99   7.99   1.13    1.13    .50     .50
                                     8.50   8.50   1.50    1.50    .59     .59
                                    16.00  16.00   1.78    1.78
                                    16.99  16.99

  *  MANUFACTURER: (W)Wemco, (A)Agitair, (O) Outokumpu, (D)Dorr
```

```
****** RESULTS : SUMMARY BALANCE ******


                     CIRCUIT : ROUGHER - SCAVENGER

                               FLOWRATE t/h
                     New Feed     Rougher Conc     Scav. Conc       Tail
Solid                150.00          20.22            12.73        129.78
Water                207.14          23.48            20.75        190.99
Water Addition                       10.06             7.33
Water adjusment                        .00

                         GRADE  and DISTRIBUTION %
                 New Feed        Rougher Conc       Scav. Conc          Tail
              Grade    Dist     Grade    Dist     Grade    Dist     Grade    Dis
CHALCOPYR       5.0   100.0      34.0    91.5       7.2    12.2        .5     8.
SPHALERIT       5.0   100.0       2.9     7.7       4.1     6.9       5.3    92.
PYRITE         20.0   100.0      11.4     7.7      16.3     6.9      21.3    92.
SILICATES      70.0   100.0      51.8    10.0      72.4     8.8      72.8    90.
Water          58.0   100.0      53.7    11.3      62.0    10.0      59.5    92.
Solid          42.0   100.0      46.3    13.5      38.0     8.5      40.5    86.



****** RESULTS : SUMMARY BALANCE ******


                     CIRCUIT : 1 st CLEANER  -SCAVENGER

                               FLOWRATE t/h
                       Feed       Cleaner 1 Conc     Scav Conc        Tail
Solid                24.5948         11.1932          .8417         12.5599
Water                57.3879         16.7363         1.6607         38.9909
Water Addition                        4.4240          .3408
Water adjusment                       6.1646

                         GRADE and DISTRIBUTION %
                   Feed         Cleaner 1 Conc      Scav Conc          Tail
              Grade    Dist     Grade    Dist     Grade    Dist     Grade    Di
CHALCOPYR      30.6   100.3      60.7    90.6      47.2     5.3       2.6     4
SPHALERIT       2.9     9.6       1.4     2.1       1.9      .2       4.4     7
PYRITE         11.7     9.6       5.6     2.1       7.7      .2      17.5
SILICATES      54.8    12.8      32.3     3.4      43.2      .3      75.6
Water          70.0    27.7      59.9     8.1      66.4      .8      75.6    1
Solid          30.0    16.4      40.1     7.5      33.6      .6      24.4
```

```
****** RESULTS : SUMMARY BALANCE ******


                  CIRCUIT : 2nd CLEANER

                                  FLOWRATE t/h
                   Feed        Clean2 Conc         Tail
Solid             12.580          9.002           3.578
Water             29.353         13.543          15.810
Water Addition                    3.257
Water adjusment                  "3.278

                         GRADE and DISTRIBUTION %
                   Feed          Clean2 Conc         Tail
              Grade    Dist    Grade    Dist    Grade    Dist
CHALCOPYR      54.9    92.1     73.7    88.5      7.6     3.6
SPHALERIT       1.6     2.7       .8     1.0      3.6     1.7
PYRITE          6.4     2.7      3.2     1.0     14.4     1.7
SILICATES      37.1     4.4     22.3     1.9     74.4     2.5
Water          70.0    14.2     60.1     6.5     81.5     7.6
Solid          30.0     8.4     39.9     6.0     18.5     2.4



****** RESULTS : SUMMARY BALANCE ******


                  CIRCUIT : 3rd CLEANER

                                  FLOWRATE t/h
                   Feed        Clean3 Conc         Tail
Solid              9.002          7.604           1.398
Water             23.149         11.616          11.532
Water Addition                    2.723
Water adjusment                   6.348

                         GRADE and DISTRIBUTION %
                   Feed          Clean3 Conc         Tail
              Grade    Dist    Grade    Dist    Grade    Dist
CHALCOPYR      73.7    88.5     85.8    87.0      8.3     1.5
SPHALERIT        .8     1.0       .4      .4      3.1      .6
PYRITE          3.2     1.0      1.5      .4     12.5      .6
SILICATES      22.3     1.9     12.4      .9     76.2     1.0
Water          72.0    11.2     60.4     5.6     89.2     5.6
Solid          28.0     6.0     39.6     5.1     10.8      .9
```

```
                ******  RESULTS : DETAILED BALANCE  ******


                      CIRCUIT: ROUGHER - SCAVENGER

                      Cumulative Distribution (%)
     Cel  CHALCOP SPHALER PYRITE  SILICAT  WATER   SOLID
      1    30.0     2.2     2.2     2.9     3.3     4.1
      2    60.0     4.5     4.5     5.8     6.6     8.2
      3    82.1     6.4     6.4     8.3     9.4    11.5
      4    91.5     7.7     7.7    10.0    11.3    13.5

      1     4.4     1.1     1.1     1.4     1.6     1.5
      2     6.9     2.1     2.1     2.7     3.1     2.8
      3     8.5     3.1     3.1     3.9     4.5     4.0
      4     9.8     4.1     4.1     5.2     5.9     5.1
      5    10.7     5.0     5.0     6.4     7.3     6.3
      6    11.5     6.0     6.0     7.6     8.6     7.4
      7    12.2     6.9     6.9     8.8    10.0     8.5


                         Cumulative Grade (%)
     Cel  CHALCOP SPHALER PYRITE  SILICAT  WATER   SOLID
      1    36.7     2.7    10.9    49.7    52.7    47.3
      2    36.6     2.7    10.9    49.7    52.7    47.3
      3    35.7     2.8    11.1    50.4    53.0    47.0
      4    34.0     2.9    11.4    51.8    53.7    46.3

      1    15.0     3.7    14.9    66.4    60.0    40.0
      2    12.5     3.8    15.3    68.3    60.7    39.3
      3    10.8     3.9    15.6    69.7    61.1    38.9
      4     9.5     4.0    15.9    70.6    61.4    38.6
      5     8.6     4.0    16.1    71.3    61.7    38.3
      6     7.8     4.1    16.2    71.9    61.8    38.2
      7     7.2     4.1    16.3    72.4    62.0    38.0
```

CIRCUIT: 1st CALENER - SCAVENGER

Cumulative Distribution (%)

Cel	CHALCOP	SPHALER	PYRITE	SILICAT	WATER	SOLID
1	40.0	.9	.9	1.5	3.5	3.3
2	71.4	1.6	1.6	2.7	6.4	5.9
3	84.9	2.0	2.0	3.2	7.6	7.0
4	90.6	2.1	2.1	3.4	8.1	7.5
1	2.7	.1	.1	.1	.3	.2
2	4.2	.1	.1	.2	.5	.4
3	5.3	.2	.2	.3	.8	.6

Cumulative Grade (%)

Cel	CHALCOP	SPHALER	PYRITE	SILICAT	WATER	SOLID
1	61.1	1.4	5.5	32.0	59.9	40.1
2	60.9	1.4	5.6	32.2	59.9	40.1
3	60.8	1.4	5.6	32.2	59.9	40.1
4	60.7	1.4	5.6	32.3	59.9	40.1
1	57.8	1.5	6.2	34.5	61.2	38.8
2	51.8	1.8	7.1	39.4	64.3	35.7
3	47.2	1.9	7.7	43.2	66.4	33.6

CIRCUIT: 2nd CLEANER

Cumulative Distribution (%)

Cel	CHALCOP	SPHALER	PYRITE	SILICAT	WATER	SOLID
1	40.0	.4	.4	.8	2.9	2.7
2	66.3	.7	.7	1.4	4.9	4.5
3	79.1	.8	.8	1.7	5.8	5.4
4	84.7	.9	.9	1.8	6.2	5.7
5	87.2	.9	.9	1.9	6.4	5.9
6	88.5	1.0	1.0	1.9	6.5	6.0

Cumulative Grade (%)

Cel	CHALCOP	SPHALER	PYRITE	SILICAT	WATER	SOLID
1	74.3	.8	3.1	21.9	60.0	40.0
2	74.0	.8	3.1	22.1	60.0	40.0
3	73.9	.8	3.2	22.1	60.0	40.0
4	73.9	.8	3.2	22.2	60.0	40.0
5	73.8	.8	3.2	22.2	60.0	40.0
6	73.7	.8	3.2	22.3	60.1	39.9

CIRCUIT: 3rd CLEANER

Cumulative Distribution (%)

Cel	CHALCOP	SPHALER	PYRITE	SILICAT	WATER	SOLID
1	40.0	.2	.2	.4	2.5	2.3
2	68.8	.3	.3	.7	4.3	4.0
3	80.6	.3	.3	.8	5.1	4.7
4	84.8	.4	.4	.9	5.3	4.9
5	86.3	.4	.4	.9	5.5	5.0
6	87.0	.4	.4	.9	5.6	5.1

Cumulative Grade (%)

Cel	CHALCOP	SPHALER	PYRITE	SILICAT	WATER	SOLID
1	86.3	.3	1.4	12.0	60.0	40.0
2	86.1	.4	1.4	12.1	60.0	40.0
3	86.1	.4	1.5	12.1	60.0	40.0
4	86.0	.4	1.5	12.1	60.0	40.0
5	86.0	.4	1.5	12.2	60.1	39.9
6	85.8	.4	1.5	12.4	60.4	39.6

ATTACHMENT X

```
                 SELECT ONE OPTION THEN ESC

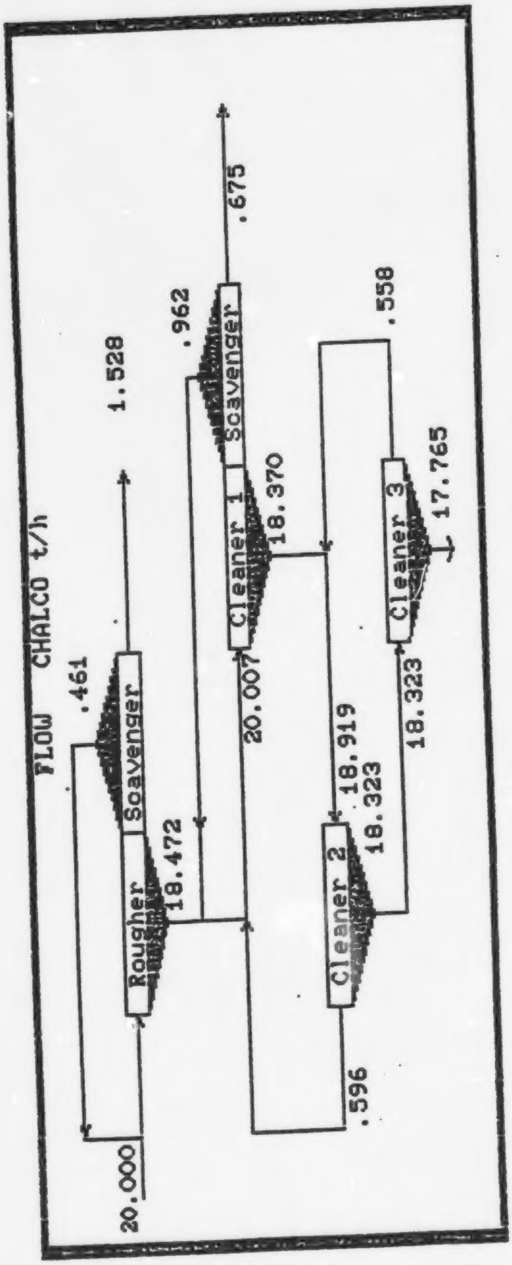

                      DATA INPUT
                      CALCULATION
                ==>   PRINT
                      READ FILE
                      SAVE FILE
                      ERASE FILE
                      EXIT
```

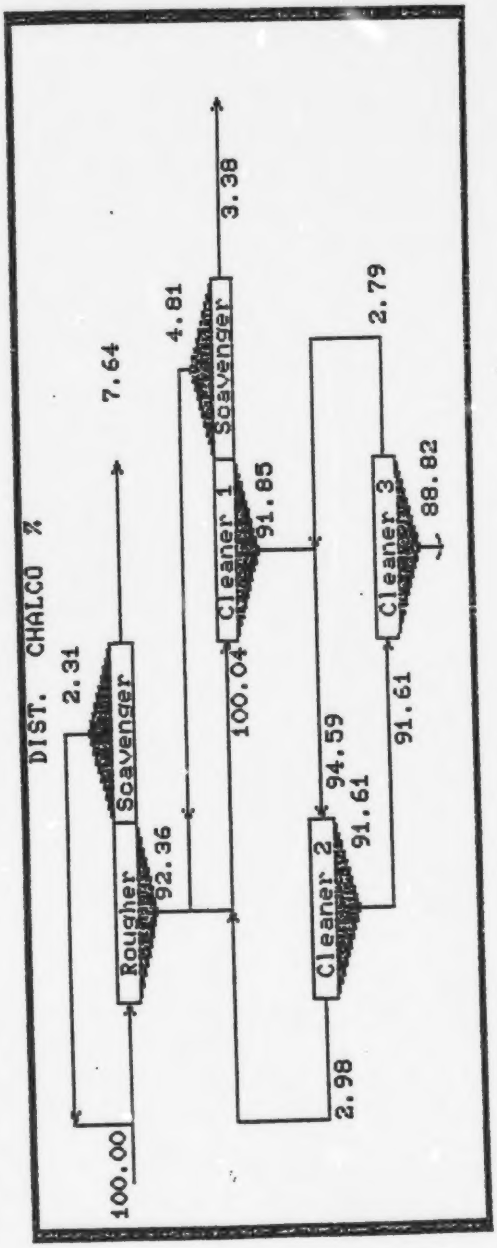
```
                       ******   PRINT   ******

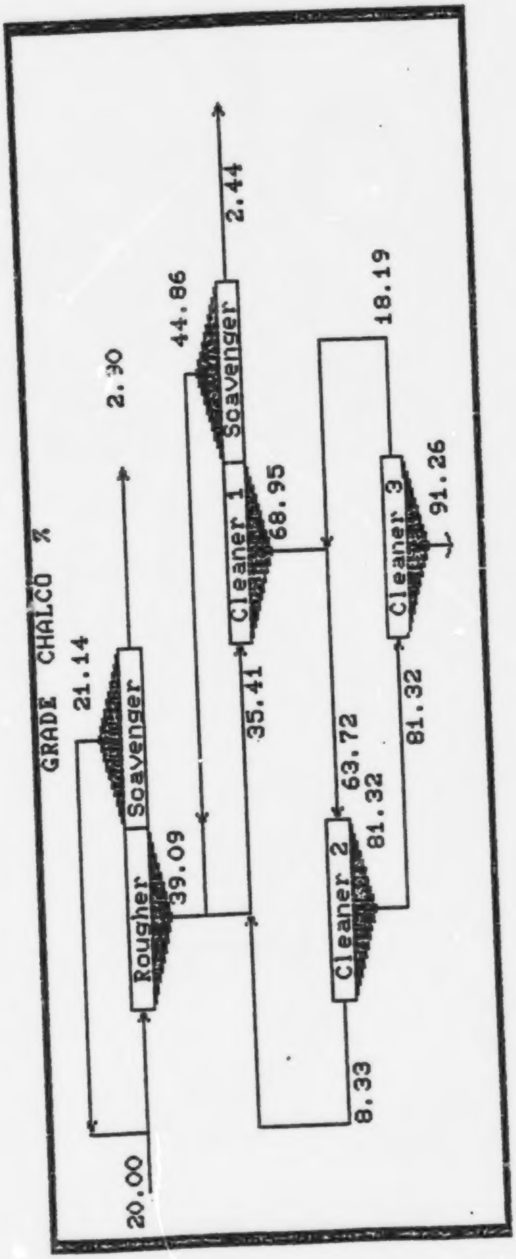
                                  Select Flowsheet Informations
PRINT DESTINATION                     Flow       ==>CHALCOPYR
     Screen                       ==> Dist          SPHALERIT
 ==> Printer                          Grade         PYRITE
                                                    SILICATES
PRINTING CONTENT                                    WATER
     Input Data                                    SOLID
     Rsults: Equipment
     Rsults: Balance
 ==> All

BALANCE FORMAT
     Simplified Table
     Detailed Table
 ==> Graphics (with Specified Graphic Card)

GRAPHIC CONTENT
 ==> Flowsheet with Flowrate, Grade or Dist.
     Curve of GRADE-DISTRIBUTION vs CELL
```


FLOW CHALCO t/h
20.000
Rougher
Scavenger
.461
18.472
1.528
Cleaner 1
Scavenger
.962
.675
20.007
18.370
Cleaner 2
18.919
18.323
.596
Cleaner 3
18.323
17.765
.558


DIST. CHALCO %
100.00
Rougher
Scavenger
2.31
92.36
7.64
Cleaner 1
Scavenger
4.81
3.38
100.04
91.85
Cleaner 2
94.59
91.61
2.98
Cleaner 3
91.61
88.82
2.79


GRADE CHALCO %
20.00
Rougher
39.09
Scavenger
21.14
2.30
Scavenger
44.86
2.44
Cleaner 1
35.41
68.95
Cleaner 2
63.72
81.32
8.33
Cleaner 3
81.32
91.26
18.19

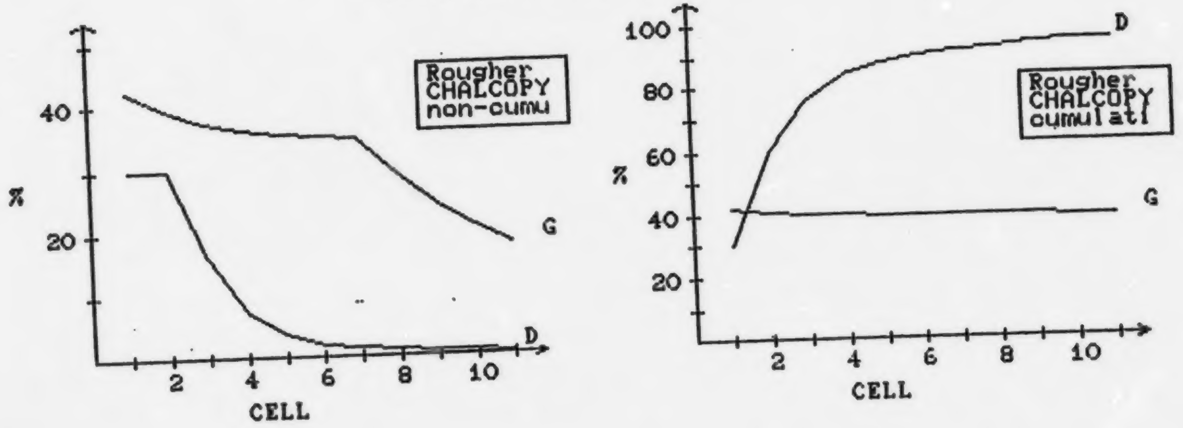
```
                    ******   PRINT   ******

                                      Select GRADE-DISTRIBUTION Curves
PRINT DESTINATION                         Rougher     ==>CHALCOPYR
      Screen                          ==> Cleaner 1      CALCITE
 ==> Printer                              Cleaner 2      PYRITE
                                          Cleaner 3      WATER
PRINTING CONTENT                                         SOLID
      Input Data
      Résults: Equipment
 ==> Résults: Balance
      All

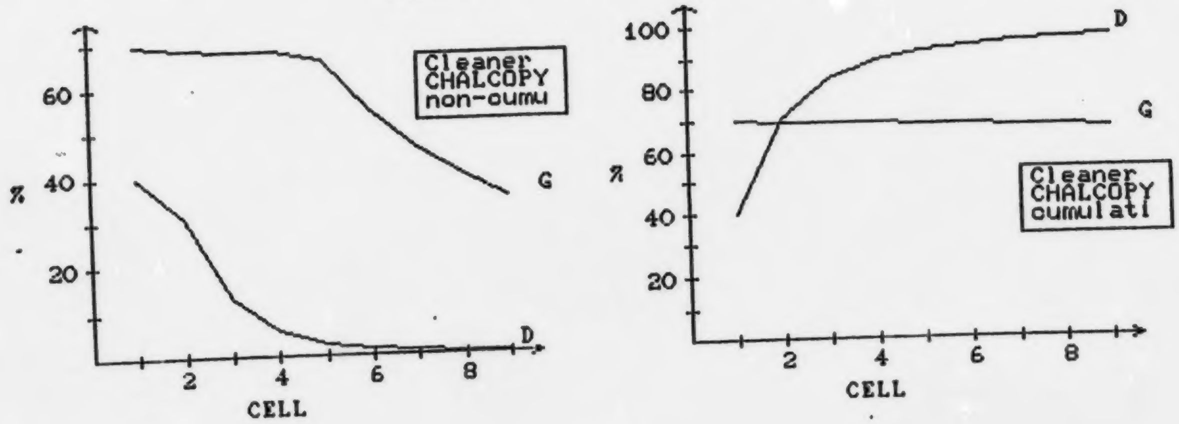
BALANCE FORMAT
      Simplified Table
      Detailed Table
 ==> Graphics (with Specified Graphic Card)

GRAPHIC CONTENT
      Flowsheet with Flowrate, Grade or Dist.
 ==> Curve of GRADE-DISTRIBUTION vs CELL
COURBE 2      ==> non-cumul.
              ==> cumulative
```

Distribution (D) and Grade (G) VS Cell



Distribution (D) and Grade (G) VS Cell



ATTACHMENT XI

CALFLOT 2.0

Logiciel de calcul de circuits de flottation

Michel BOURASSA
Gilles BARBERY

Rapport GRAIIM 87-4
Août 1987

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Michel Bourassa, P. Eng.

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1. INTRODUCTION

Les usines minéralurgiques renferment plusieurs appareils jouant chacun un rôle particulier. Le dimensionnement de ces équipements unitaires se réalise assez bien car, il y a pour la plupart des cas une ou plusieurs procédures reconnues. Certaines procédures ne nécessitent pas ou très peu d'essais de laboratoire (exemple de la méthode de Krebs pour les hydrocyclones) mais la grande majorité l'exige. Bien que l'empirisme gouverne la plus grande part des procédures (les méthodes de Tyler et Allis Chalmers pour les tamis), certaines ont un caractère plus phénoménologique (Talmage et Fitch pour les épaississeurs, méthode de Bond, la loi de Poiseuille et la filtration).

La flottation demeure aujourd'hui un des procédés où il n'y existe pas de procédures reconnues pour le dimensionnement. Il y a certaines règles énoncées mais elles ne sont pas utilisées en pratique. Ceci est probablement lié aux problèmes rencontrés lors de la prédiction de la cinétique du procédé.

Le logiciel CALFLOT (version 2.0) incorpore les plus importantes règles de dimensionnement à un simulateur (en mode de prédiction) et permet de dimensionner les usines de flottation. L'utilisation de CALFLOT permet de calculer les caractéristiques des équipements comme la configuration du circuit, le nombre et le volume des cellules, le nombre de bancs de cellules ainsi que le bilan minéralurgique complet. La version CALFLOT 1.0 ne permettait que le calcul d'un banc de flottation et on ne pouvait pas y flotter deux minéraux de valeur (hydrophobes) simultanément.

2. DOMAINE D'APPLICATION

Pour bien comprendre le champ d'application de CALFLOT, il faut faire une nuance entre un simulateur et un logiciel de dimensionnement. Un simulateur permet de reproduire ou prédire le comportement d'un appareil (ou groupe d'appareils) dans un environnement déterminé à l'avance. Un logiciel de dimensionnement, comme CALFLOT, utilise généralement un simulateur en mode de prédiction pour déterminer l'environnement physique, c'est à dire le ou les appareils nécessaires. La sélection des équipements se fait automatiquement en suivant les régles et procédures de dimensionnement. Afin de visualiser la structure et les possibilités de CALFLOT, on abordera séparément la partie simulateur et la partie dimensionnement.

Le dimensionnement d'un circuit de flottation consiste à déterminer les quatre éléments suivants:

1- configuration du circuit:	ceci comprend le nombre d'étapes de concentration ainsi que la façon dont sont reliés les différents flux.
2- le volume des cellules:	le volume des cellules pour toutes les étapes de concentration.
3- le nombre de cellules:	le nombre de cellules pour chaque étape de concentration.
4- le nombre de bancs:	le nombre de bancs de cellules disposés en parallèle pour chaque étape de concentration.

CALFLOT permet de déterminer l'ensemble de ces quatre éléments ou un ensemble plus restreint, selon que l'on désire fixer certaines parties du circuit. On fera un dimensionnement nul si tous les éléments sont connus (c.a.d. fixés par l'utilisateur), un dimensionnement complet s'ils sont tous inconnus et un dimensionnement partiel dans les autres cas (tab. 1).

Le simulateur statique intégré à CALFLOT a une fonction de prédiction de bilan. Le bilan obtenu est utilisé par la suite pour déterminer les éléments du dimensionnement. Un simulateur est un ensemble d'équations ou de modèles mathématiques. Il existe plusieurs simulateurs de flottation et il se différencient tant par leur nombre de modèles que par la complexité de ceux-ci. CALFLOT utilise trois modèles primaires soit un pour les minéraux hydrophobes, un pour les minéraux hydrophiles et un pour l'eau. Qui dit modèles, dit paramètres et calibrage de ceux-ci. Les paramètres d'un modèle ne sont pas nécessairement connus par l'utilisateur. Si ce dernier possède une valeur numérique pour l'ensemble des paramètres des trois modèles, il a une connaissance complète des modèles. A l'opposé, il aura une connaissance nulle des modèles s'il n'a aucune valeur numérique pour les paramètres et une connaissance partielle dans les autres cas (tab. 1).

Le champ d'application général de CALFLOT est obtenu en sélectionnant toutes les combinaisons possibles entre le type de dimensionnement et l'état de connaissance du simulateur. Il est à remarquer qu'un simulateur est généralement la combinaison:

"(A):aucun dimensionnement" et "(3):connaissance complète des modèles"

A l'opposé, la combinaison:

"(C):dimensionnement complet" et "(1):connaissance nulle des modèles"

s'avère très intéressante pour le dimensionnement. CALFLOT possède des limitations (Ex.: le nombre de configurations possibles) et c'est pour être plus clair que nous les omettons à ce niveau-ci. Les limitations seront exposées plus loin dans le texte.

DIMENSIONNEMENT		SIMULATEUR	
Nombre d'éléments à dimensionner	Type	Nombre de modèles déterminés	connaissance du modèle
0	(A) aucun	0	(1) aucune
1-2-3	(B) partiel	1-2	(2) partielle
4	(C) complet	3	(3) complète

Tableau 1: Représentation du champ d'application de CALFLOT

3. METHODES DE DIMENSIONNEMENT

Pour chacun des quatre éléments de dimensionnement énumérés précédemment, il existe des critères de sélection. Dans cette section, nous allons les décrire de manière générale en mettant l'emphase sur ceux utilisés dans le logiciel CALFLOT.

3.1 Configuration des circuits

Pour la production d'un type de concentré, il peut y avoir un nombre important de configurations possibles. La teneur, le degré de flottabilité sont des variables qui doivent être considérées lorsqu'il faut fixer la configuration d'un circuit. Il y a aussi des facteurs plus arbitraires comme: la stratégie de contrôle envisagée (commande) et le jugement ou la perception des personnes affectées à cette tâche. Comme chaque minéral (ou groupe de minéraux) a des exigenges particulières, le nombre total de configurations possibles est trop grand pour qu'elles soient toutes considérées. On a décidé de rationaliser le nombre de configurations en se limitant à celles qui se rencontrent le plus souvent. Dans CALFLOT, on a limité à quatre le niveau de concentration (tab. 2). A quelques exceptions près (molybdénite), ceci est suffisant pour les concentrés produits par flottation. Aux deux premiers niveaux, on a la possibilité d'ajouter une étape d'épuiseurs. Ce cadre est assez rigide mais il permet néanmoins de couvrir les applications les plus courantes (Malgham 1986). Une fois le cadre délimité, le problème de dimensionnement consiste à trouver le nombre de niveaux et le mode d'agencement de tous les flux qui seront nécessaires pour produire le concentré ayant la teneur voulue. Le premier niveau permettant une teneur en minéral enrichi supérieure ou égale à celle désirée devient le niveau final.

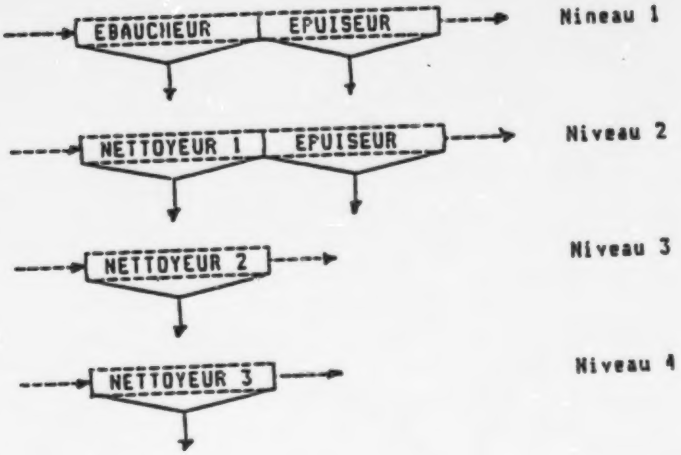

__Tableau 2:__ Structure de base des configurations

La recirculation des pulpes en amont des circuits de flottation a tendance à les stabiliser (Bourassa 1986). Cette pratique, jadis fréquente, semble en perte de vitesse et on mise maintenant davantage sur la commande pour stabiliser les circuits. Ceci ne veut pas dire qu'il n'est pas utile de recirculer les pulpes en amont. Au contraire, ceci est bénéfique en certaines occasions (Sutherland 1981) mais il faut faire subir un traitement aux flux recirculés. De récents travaux (Jowett et Sutherland 1985) ont permis de confirmer ce qui était autrefois mentionné (Taggart 1945) de façon intuitive, à savoir: lorsqu'il y a recirculation de flux en amont du circuit de flottation, il faut ajouter ce ou ces flux à un endroit où la composition minérale et la teneur en eau y sont voisines. La flottabilité des produits recirculés est généralement considérée égale à celle de la pulpe alimentant le banc (Agar et Kipkie 1978). Cette hypothèse n'est pas juste mais l'erreur introduite est faible car le débit relatif du flux recirculé est peu important. On présente ci-dessous les règles utilisées par CALFLOT pour fixer la configuration du circuit. Il se peut que ces règles ne soient que partiellement employées selon que l'utilisateur impose en tout ou en partie la configuration du circuit. Cesrègles sont:

- la partie des nettoyeurs est complètement séparée de celle des ébaucheurs (aucun retour des nettoyeurs aux ébaucheurs),
- la recirculation des pulpes en amont du circuit ne peut se faire qu'au même niveau ou à celui qui le précède,
- la présence d'épuiseurs est justifiée si l'on peut produire un concentré d'épuiseurs ayant une teneur inférieure à celle de l'alimentation du banc,
- la teneur en minéral enrichi dans le flux recirculé doit s'approcher, le plus possible, de celle du flux cible,
- le concentré (principal s'il y un épuiseur) du niveau "i" alimente le niveau "i+1", et
- les rejets des niveaux 1 et 2 sont des rejets finaux.

Un premier calcul en circuit ouvert permet de fixer la configuration qui demeure inchangée pour le reste des calculs. Cette configuration n'est pas optimale mais elle est satisfaisante compte tenu de la qualité des modèles de prédiction disponibles à ce jour. Des critères d'optimisation pourront éventuellement être ajoutés lorsque les modèles de prédiction seront plus performants. Parmi les travaux faits dans ce domaine on peut citer celui de Sutherland (1981) qui montre que, pour un circuit à deux niveaux, l'agencement optimal s'obtient en présence d'épuiseurs (aux ébaucheurs) lorsque le temps de séjour aux ébaucheurs est environ le même qu'aux nettoyeurs. Plusieurs auteurs énoncent une méthode générale pour optimiser l'agencement des flux (Mehrotra et Kapur 1974, King 1976, Neimi et al. 1982). Ces méthodes ont le désavantage d'être très lourdes lorsque le nombre de cellules est important.

3.2 Volume des cellules de flottation

Il y a plusieurs types de cellules de flottation. Les cellules agitées mécaniquement, les plus employées, sont celles qui nous intéressent. Il y a.

parmi ce groupe, de nombreux produits se différenciant soit par le mode d'injection d'air, par le mode d'agitation de la pulpe ou par le mode d'évacuation des mousses. On peut trouver ailleurs une bonne description des cellules disponibles (Harris, 1976, Poling, 1980, Barbery, 1984) ainsi que le type d'application suggéré (certaines cellules sont conçues pour des applications spécifiques). On a inclus dans CALFLOT une banque de données contenant les caractéristiques des cellules de quatre grands fabricants soit: Outokumpu, Wemco, Dorr et Agitair. Certaines de ces cellules sont de type simple (un agitateur, une cuve et un moteur) d'autres sont de type double (deux agitateurs, deux cuves et un moteur). Le volume considéré lors de la sélection est celui d'une cuve qu'elle soit séparée ou non des cellules connexes par des compartiments.

La méthode conventionnelle de sélection des cellules (Poling, 1980) considère simultanément le volume unitaire, le nombre d'unités ainsi que le nombre de bancs. On détermine en premier lieu le volume total de flottation nécessaire pour obtenir un rendement de référence. A ce rendement de référence, généralement obtenu à partir d'essais de laboratoire, on associe un temps de flottation. Sur le temps obtenu, on applique un facteur d'échelle pour déterminer le temps industriel de flottation. Comme l'ont fait état certains travaux (Barbery et al, 1986, Spada, 1981), il y a une grade diversité dans les facteurs d'échelle proposés qui se situent entre 1.5 et 4.

Par la suite, on fixe le volume nominal des cellules ainsi qui le volume effectivement occupé par la pulpe (50 à 85% du volume nominal). Le choix du volume nominal semble très arbitraire dans certains cas (Poling, 1980). Faute de mieux, on peut utiliser les tables des fabricants qui donnent une capacité (t/j de solides) par unité de volume des cellules. Ces tables ne doivent servir que pour une grossière approximation du volume des cellules car il y a des facteurs importants (comme la concentration en solides) qui n'interviennent pas dans la sélection (Arbiter et al, 1980). La sélection du volume des cellules a été rendue moins arbitraire par l'introduction de données supplémentaires (Spada, 1981, Arbiter, 1986) concernant le temps de séjour unitaire (un intervalle de temps de séjour) applicable pour les minéraux d'une même famille. Ces temps de séjour, compris entre 0.7 et 3 minutes, proviennent de l'analyse des circuits industriels existants. L'utilisation de ce type de données est recommandée lorsqu'on ne dispose pas de résultats d'essais de laboratoire mais il ne faut pas perdre de vue que ce choix sera très conservateur. A titre d'exemple, une cellule aux ébaucheurs qui concentre un minéral de cuivre devrait avoir un temps de séjour unitaire de l'ordre de 0.95 min. (Arbiter 1986). Spada (1981) fourni aussi des temps de séjour unitaires pour déterminer le volume des cellules au niveau des nettoyeurs. Il propose également d'utiliser les résultats d'essais de laboratoire (lorsqu'ils sont accessibles) pour déterminer le temps de séjour unitaire. La méthode dite du T_{50} prend comme temps de séjour unitaire le temps de flottation en laboratoire nécessaire pour obtenir un rendement de 50%. Elle peut être utilisée tant aux ébaucheurs qu'aux nettoyeurs. Cette méthode semble intéressante mais il n'y a pas assez de preuves illustrant son bon fonctionnement pour en recommander l'utilisation.

Degner (1986) propose une méthode de sélection basée sur des critères économiques. Dans ce cas-ci, on sélectionne la combinaison (volume et nombre de cellules, et nombre de bancs) qui permet de minimiser des critères économiques tout en respectant un objectif de rendement. La surface occupée par le circuit de flottation ainsi que sa longueur sont les critères à minimiser. Les contraintes sont le rendement et les dimensions du circuit.

Le logiciel CALFLOT choisit le volume des cellules qui permettent d'obtenir un temps de séjour unitaire le plus près possible de 2 minutes. Ce temps de séjour unitaire est calculé en prenant le débit volumique à l'alimentation du banc. Le temps de séjour unitaire doit être dans l'intervalle 1-3 min. Pour assurer le respect de cette contrainte, il sera parfois nécessaire de modifier le nombre de bancs disposés en parallèle. Le volume effectif est fixé à 75% du volume nominal. L'intervalle 1-3 minutes est le même pour les quatre niveaux. Il est prévu, pour les développements futurs, que l'intervalle puisse varier en fonction du volume des cellules, du niveau de concentration et du minéral enrichi. Les cellules d'un même niveau (ébaucheurs et épuiseurs) ont pour l'instant toute la même dimension. Il n'est pas possible ainsi de dimensionner un circuit avec de très grosses cellules aux ébaucheurs suivies de cellules plus petites aux épuiseurs. Le volume nominal, la proportion du volume efficace ainsi que le fabricant des cellules peuvent être imposés, en tout ou en partie, à chacun des niveaux.

3.3 Nombre de cellules

Selon la méthode convetionnelle (Poling, 1980), le volume total de flottation est déterminé en multipliant le débit volumique de la pulpe à l'alimentation du niveau par le temps de flottation industriel. Le nombre total de cellules s'obtient en divisant le volume total de flottation par le volume effectif d'une cellule tel qu'obtenu précédemment. On répartit par la suite le nombre total de cellules en bancs identiques permettant chacun un temps de flottation se rapprochant du temps industriel de flottation.

D'un point de vue conçeptuel les circuits de flottation se sont modifiés au cours des dernières années. Les circuits de flottation des années soixantes étaient constitués d'un grand nombre de petites cellules agencées en bancs parallèles. A titre d'exemple, Arbiter et Weiss (1970) mentionnaient qu'il était préférable d'opter pour un circuit de 2 bancs de 14 cellules de 200 pi^3 que pour un autre n'ayant qu'un banc de 18 cellules de 300 pi^3. Aujourd'hui, on suggère d'utiliser des cellules beaucoup plus grosses et de minimiser le nombre d'unités (Arbiter, 1980). L'utilisation des grosses cellules permet de diminuer considérablement les coûts de production. Ceci a été confirmé par plusieurs chercheurs (Degner, 1986, Jowett et Sutherland, 1979) qui font toutefois mention de l'importance de garder un nombre minimal de cellules en série. Un trop petit nombre de cellules en série occasionne une diminution du rendement. Cette baisse est alors imputée au court circuitage d'une partie de la pulpe dans le banc de flottation. La baisse peut théoriquement être compensée par l'augmentation du volume des cellules (Jowett et Sutherland 1979) mais il y a des limites pratiques qui ne peuvent être franchies. Le nombre minimal de cellules en série varie selon le minéral enrichi mais en aucun cas il ne devrait être inférieur à 4 (Arbiter, 1986, Degner, 1986).

Agar et al. (1980) ont présenté trois critères permettant de sélectionner le nombre de cellules en série. Ces critères se différencient selon l'approche mais donnent essentiellement les mêmes résultats. Le temps optimal de flottation est obtenu soit:
- lorsque la différence de rendement entre le minéral de valeur et les minéraux de gangue est maximale,
- lorsque la cinétique de flottation du minéral de valeur et celle de la gangue sont identiques, et
- lorsque la teneur marginale du concentré est la même que celle de l'alimentation du banc.

Cette méthode a été appliquée avec succès au niveau des ébaucheurs (Kalapudas, 1985, Agar, 1985) et au niveau des nettoyeurs (Agar, 1985). Selon cette procédure, la teneur des rejets aux ébaucheurs est généralement trop élevée pour qu'ils soient acheminés directement aux rejets finaux. La présence d'épuiseurs permet d'abaisser cette teneur à un niveau raisonnable. Le temps de flottation aux épuiseurs peut être obtenu avec la même méthode en considérant le rejet des ébaucheurs comme étant l'alimentation d'une nouvelle étape de concentration. Cette approche nécessite la prédiction du comportement des minéraux hydrophobes (minéraux de valeur) mais également celle des minéraux hydrophiles (de gangue).

Pour fixer le nombre de cellules par banc, le logiciel CALFLOT 2.0 utilise les trois critères suivants:

- le rendement marginal doit être supérieur à une valeur limite,
- la teneur marginale du concentré doit être supérieure à la teneur d'alimentation de la cellule, et
- la teneur globale du concentré doit être supérieure à la teneur désirée.

Le coût de production (opération, investissement, traitement et transport) supplémentaire doit être inférieur au revenu (incluant un amortissement de l'investissement) engendré par le rendement marginal que procure cette cellule. L'utilisateur peut fixer le rendement marginal permis, et ce, à tous les niveaux. Une cellule qui donne un concentré ayant une teneur plus faible que celle qu'on y retrouve à l'alimentation n'est évidemment pas justifiée. Ce critère a davantage une fonction sécuritaire qu'une fonction de dimensionnement. Le dernier critère permet à l'utilisateur d'imposer la teneur du concentré à chacun des niveaux s'il le désire. Ce critère peut être utilisé lorsque la teneur du concentré produit est plus importante que le rendement obtenu. Le nombre de cellules, pour chacun des niveaux, est fixé en circuit ouvert par le premier des critères atteints.

3.4 Nombre de bancs

Le nombre de bancs n'est pas une composante qui peut être sélectionnée seule. Le nombre de bancs est toujours évalué en tenant compte des autres éléments comme le volume des cellules et le nombre de cellules. Des chercheurs ont démontré (Jowett et Sutherland, 1979, Degner, 1986) que, d'un point de vue économique, il faut minimiser le nombre de bancs disposés en parallèle. Cette approche est intégrée dans CALFLOT 2.0 et ce n'est qu'exceptionnellement que le logiciel choisi plus d'un banc. Ceci se produit lorsque les plus grosses cellules disponibles ne permettent pas un temps de séjour unitaire supérieur à la limite inférieure (1 min).

3.5 Sélection du fabricant

Le dimensionnement des circuits de flottation est fait sans accorder d'importance à la marque des cellules qui sera sélectionnée par la suite. Il y a certainement des différences d'une marque à l'autre mais celles-ci n'ont pas été caractérisées au niveau de la cinétique de flottation. La sélection du fabricant est généralement basée sur la réputation des produits qu'il offre ou sur des points techniques particuliers (Arbiter et Weiss, 1970). Les coûts d'investissement, les coûts d'opération, le mode d'aération, le contrôle du niveau de pulpe et la qualité du service offert sont autant de facteurs qui peuvent être déterminants dans le choix du fabricant. Le logiciel CALFLOT 2.0 donne une liste des fabricants qui fournissent les cellules sélectionnées. L'utilisateur peut toutefois imposer que la sélection des cellules se fassent dans la gamme offerte par un des fabricants.

4. PREDICTION ET MODELES MATHEMATIQUES

Le simulateur intégré à CALFLOT 2.0 regroupe trois modèles. Ces modèles permettent de simuler le comportement des trois composantes principales qui sont: les minéraux hydrophobes ou de valeur, les minéraux hydrophiles ou de gangue et l'eau. Leur structure diffère selon qu'ils s'appliquent à la flottation continue ou semi-continue. Lorsqu'on désire extrapoler les résultats obtenus en laboratoire au niveau industriel, il faut modifier les modèles pour qu'ils tiennent compte de l'effet d'échelle. Des facteurs d'extrapolation appliqués sur la cinétique de flottation en laboratoire rendent possible la prédiction. Ces facteurs d'échelle sont rarement disponibles et, même lorsqu'ils le sont, il peut arriver que les résultats prédits soient irréalisables dans la pratique.

Dans cette section nous allons décrire sommairement, pour chacun des constituants, le modèle utilisé en laboratoire ainsi que celui dérivé pour la flottation industrielle. On discutera également des facteurs d'extrapolation et des limitations propres à chacun des modèles. Les modèles étant les mêmes que ceux utilisés dans la version 1.0, la présentation a été simplifiée et on pourra consulter le premier manuel pour des renseignements supplémentaires (Bourassa et Barbery 1985).

4.1 Minéraux de valeur (hydrophobes)

La cinétique de flottation des minéraux hydrophobes dans une cellule de laboratoire est représenté par:

$$R(t) = R_{\infty}[1-(1-\varsigma)e^{-K_f t} + \varsigma e^{-K_s t}] \qquad \text{(éq. 1)}$$

Cette équation a été dérivée selon l'hypothèse d'une cinétique d'ordre 1 et d'une distribution discrète (3 populations) des constantes cinétiques. Le rendement (R) est alors fonction du temps (t) et de 4 paramètres qui sont:

K_f: constante cinétique de la population flottant rapidement,
K_s: constante cinétique de la population flottant lentement,
ζ : proportion de la population active qui flotte lentement,et
R_∞: rendement ultime.

Le passage de la flottation semi-continue à la flottation continue est fait par l'intermédiaire de la loi de distribution des temps de séjour des cellules alimentées en continu. Avec l'hypothèse que cette loi est celle d'un mélangeur parfait (éq. 2), on dérive l'équation (éq. 3) qui permet de calculer le rendement au terme de la première cellule. Le temps de séjour moyen (τ) est obtenu en divisant le volume efficace de la cellule par le débit volumique des rejets.

$$E(t) = \frac{1}{\tau} e^{-t/\tau} \qquad \text{(éq. 2)}$$

$$R(\tau) = R_\infty \left[\frac{(1-\zeta)K_f\tau}{1+K_f\tau} + \frac{\zeta K_s\tau}{1+K_s\tau}\right] \qquad \text{(éq. 3)}$$

En généralisant l'équation 3, on obtient une relation qui permet de calculer le rendement cumulé en fonction du nombre de cellules en série. On a après "N" cellules:

$$R_N = R_\infty \left[(1-\zeta) \sum_{j=1}^{N} \frac{K_f\tau_j}{\prod_{i=1}^{j} 1+K_f\tau_i} + \zeta \sum_{j=1}^{N} \frac{K_s\tau_j}{\prod_{i=1}^{j} 1+K_s\tau_i} \right] \qquad \text{(éq. 4)}$$

L'extrapolation des résultats de laboratoire à l'échelle industrielle est faite en appliquant des facteurs d'échelle sur les constantes cinétiques. Cette façon de procéder permet de résoudre le problème soulevé par Kalapudas (1985) à l'effet que le facteur d'échelle appliqué sur le temps de flottation en laboratoire varie selon le rendement considéré. On utilise dans CALFLOT 2.0 deux facteurs d'échelle qui s'appliquent sur les constantes K_f et K_s. Des résultats obtenus sur des minéraux sulfurés (Bourassa, 1986, Bourassa et al, 1987) indiquent que le facteur d'échelle sur K_s varie entre 1 et 5 alors que celui affectant K_f varie entre 3 et 11. Dans la gamme étudiée, les facteurs varient de manière croissante avec le volume des cellules. Ces données sont incomplètes et des travaux en cours devraient permettre d'élargir les connaissances à ce sujet. Les deux autres paramètres ne semblent pas affectés significativement par l'effet d'échelle (Bourassa, 1986). Loveday et Marchant (1972) ont utilisé avec succès le rendement ultime (R_∞) évalué en laboratoire pour simuler un circuit industriel de flottation.

L'utilisateur qui ne possède pas de résultats de laboratoire ou de facteurs d'échelle peut quand même utiliser CALFLOT 2.0 en jumelant le minéral concerné avec un minéral de référence. Le logiciel utilise dans ce cas les résultats publiés par Spada (1981). Le minéral aura un comportement moyen par rapport aux mineraux de même nature.

Au modèle décrit précédemment, on a ajouté la possibilité de contrôler le rendement maximal qu'une cellule peut procurer. Cette contrainte modifie la cinétique initiale sans altérer la structure du modèle. Son utilisation permet d'obtenir des prédictions plus conformes à la réalité et devient indispensable lorsque la cinétique est mal déterminée.

La cinétique des produits recirculés n'est pas nécessairement la même que celle du flux principal. L'utilisateur peut ainsi modifier la cinétique en changeant la proportion du minéral qui flotte lentement (ζ). Les constantes cinétiques ne peuvent pas être changées et elles ont toujours la même valeur que celle du flux principal. Si l'utilisateur ne connait pas la cinétique du produit recirculé, on fait l'hypoyhèse qu'il a la même cinétique que le flux principal.

4.2 Minéraux de gangue (hydrophiles)

La structure employée pour simuler le transfert des minéraux de gangue est inspirée des travaux effectués par Lynch (1981). En laboratoire, on suppose que le débit de gangue "i" qui passe dans le concentré (CG_i) est proportionnel au débit d'eau dans le concentré (CE) ainsi qu'à la concentration du minéral de gangue "i" dans la cellule (MG_i/ME). Le coefficient de proportionnalité se nomme le coefficient d'entrainement et se détermine bien en laboratoire en utilisant l'équation suivante:

$$CF_i = \frac{ME\ /\ CE}{MG_i/CG_i} \qquad \text{(éq. 5)}$$

Cette définition est aussi valable pour des cellules alimentées en continu. Les cellules étant été assimilées à des mélangeurs parfaits, on remplace le terme de concentration utilisant des masses (MG_i/ME) par le rapport des débits dans les rejets (RG_i/RE ou $[AG_i-CG_i]/[AE-CE]$). On obtient ainsi la relation (éq. 6) qui permet d'évaluer le débit de gangue "i" dans le concentré.

$$CG_i = \frac{CE\ AG_i\ CF_i}{AE-CE(1-CF_i)} \qquad \text{(éq. 6)}$$

Le calcul des débits de gangue devient itératif lorsque le débit d'eau dans le concentré (CE) n'est pas déterminé. Ceci est le cas, comme nous le verrons plus loin, lorsque la concentration en solides maximale contrôle le débit d'eau dans le concentré.

La valeur des coefficients d'entrainement est difficilement extrapolable dans l'état actuel des connaissances. Contrairement à ce qui se produit pour les minéraux hydrophobes, la cinétique de flottation des minéraux hydrophiles est plus lente en laboratoire (Bourassa et al, 1987). Les coefficients d'entrainement se situent entre 0.3 et 0.8 au laboratoire alors qu'ils varient de 0.4 à plus de 1 pour des cellules industrielles.

Le logiciel CALFLOT 2.0 n'utilise qu'un coefficient d'entrainement par minéral et par classe granulométrique pour l'ensemble des niveaux de concentration. Cette version n'offre pas la possibilité d'utiliser des facteurs d'échelle car il n'y a pas encore suffisamment de données disponibles. Si les coefficients ne sont pas fournis, le logiciel utilise des valeurs qui permettent l'obtention d'un comportement moyen.

4.3 Eau

L'eau est probablement la phase la plus difficile à modéliser. Le modèle considéré est celui décrit par Lynch (1981). En flottation semi-continue (au laboratoire), on assume que la masse transférée dans le concentré est proportionnelle à celle présente dans la cellule. Avec une seule constante cinétique (K_w) on obtient la relation suivante:

$$R(t) = 1 - \frac{M_E}{M_0} = 1 - e^{-K_w t} \qquad \text{(éq. 7)}$$

Comme nous l'avons fait précédemment pour les minéraux hydrophobes (de valeur), on tient compte de la distribution de temps de séjour (éq. 2) pour adapter la relation au régime continu. La relation obtenue (éq. 8) permet de calculer le rendement en eau au terme de la première cellule.

$$R(\tau) = \frac{K_w \tau}{1 + K_w \tau} \qquad \text{(éq. 8)}$$

En généralisant cette équation au cas de "N" cellules en série, on a:

$$R_N = \sum_{i=1}^{i=N} \frac{K_w \tau}{\prod_{j=1}^{j=i} (1+K_w \tau)} \qquad \text{(éq. 9)}$$

La cinétique de transfert de l'eau ne dépend que d'un paramètre (K_w) qui peut être déterminé en laboratoire. La détermination de ce paramètre devient compliquée lorsque les essais de flottation sont faits en maintenant le niveau constant par ajout d'eau. Le problème a été soulevé par Lynch (1981) et il propose de modéliser la masse d'eau recueillie en fonction du temps. Cette approche peut difficilement être appliquée pour l'extrapolation car elle n'est pas adimensionnelle.

Le modèle à une constante cinétique ne permet pas d'obtenir une représentation fidèle des résultats obtenus à l'échelle industrielle (Lynch, 1981). Un modèle à deux constantes cinétiques serait mieux adapté pour simuler le comportement de l'eau mais, comme il est déjà difficile d'évaluer un paramètre, cette possibilité n'a pas été retenue. CALFLOT 2.0 utilise un modèle d'ordre 1 avec une constante cinétique pour la prédiction de l'eau. La constante cinétique peut prendre une valeur propre à chacun des niveaux de concentration. Si l'utilisateur ne possède pas la valeur des constantes cinétiques, le logiciel opte pour une valeur donnant un comportement moyen.

Il est évident que la concentration en solides du concentré d'une cellule ne peut en pratique dépasser une certaine valeur. Cette valeur dépend de la masse volumique des minéraux flottés. A titre d'exemple, elle peut atteindre 61% pour un concentré mixte de galène et sphalérite (Forssberg et al, 1982), 55% pour un concentré de plomb (Cameron et al, 1971), probablement sous forme de galène, et 40% pour un concentré de chalcopyrite (Dionne 1986). Il est possible dans CALFLOT 2.0 de fixer la concentration en solides maximale de la pulpe dans le concentré. Ceci peut se faire de façon indépendante à tous les niveaux de concentration. Cette limitation produit, dans la plupart des cas, le même effet qu'un modèle à deux constantes cinétiques. La variation de la concentration en solides des concentrés peut alors être simulée avec plus de justesse qu'elle ne peut l'être sans la présence de cette contraite.

Lorsque la limitation sur la concentration en solides est employée, le débit d'eau dans le concentré dépend de celui des solides. Comme l'évaluation des débits de gangue nécessite la connaissance du débit d'eau (éq. 6), on doit réaliser ces calculs de manière itérative.

De l'eau est généralement ajoutée aux concentrés afin de briser l'écume et ainsi permettre un meilleur écoulement de la pulpe. Cette quantité d'eau peut être fournie par l'utilisateur ou calculée automatiquement par le logiciel. Le débit d'eau nécessaire est fixé de sorte que le rapport entre le débit volumique des solides et celui de l'eau ajoutée soit de 0.6.

5. FONCTIONNEMENT DU LOGICIEL

5.1 Généralités

Le logiciel CALFLOT 2.0 est écrit en langage APL et est disponible sur disquettes de $5^{1/4}$" et $3^{1/2}$". Son utilisation nécessite l'interpréteur APL version 5.0 ou les versions plus récentes. Le mode graphique EGA est supporté à la condition de posséder au minimum la version 6.0. Il fonctionne sur IBM PC et ceux qui sont compatibles. Avant de pouvoir l'utiliser, il faut faire l'intallation en copiant le fichier APL.EXE sur la disquette CALFLOT 2.0. Une fois l'installation complétée, le logiciel peut être chargé en suivant la procédure ci-dessous:

- charger le système d'exploitation DOS version 2.1 ou plus,
et la disquette de données dans le lecteur B,
- insérer la disquette CALFLOT 2.0 en A et une disquette de données (formatée) en B (facultatif).
- charger le logiciel en tapant <CALFLOT> suivi d'une commande de retour.

Le logiciel est convivial et le temps de calcul est relativement court. Il est possible en tout temps d'avoir accès à de l'aide en tapant simultanément les touches <CTRL> et <H> (<Ctrl-H>). On y trouve une synthèse des touches spéciales. La structure employée dans CALFLOT est sous forme de menu. Le menu principal nous invite à choisir une des options suivantes:

```
—> Acquisition des données
        Calcul
        Imprimer
        Lire
        Sauvegarder
        Effacer
        Terminer
```

Comme c'est le cas chaque fois qu'il faut choisir une option, on déplace la flèche (en utilisant les clés de déplacement du curseur) vis-à-vis l'option désirée et on valide celle-ci en appuyant sur la touche <Esc> (Escape). La touche <Esc> est aussi utilisée pour valider les fenêtres de données numériques et alphanumériques. Dans cette section, nous allons donner une brève description des options du menu principal.

5.2 ACQUISITION DES DONNEES

Dans cette option, on définit toutes les données qui seront utilisées lors du calcul du circuit. On y passe plusieurs écrans (Annexe 1) et il faut entrer à chacun les valeurs demandées. Les données qui apparaissent sur chaque écran sont les données en mémoire. Ces données restent en mémoire tant qu'elles ne sont pas effacées selon une des trois possibilités:

- en entrant des nouvelles données au clavier,
- en lisant un fichier, et
- en terminant la session.

Cette structure de mise en mémoire permet une grande flexibilité dans l'édition des données. Il y a des touches spéciales qui permettent de retourner au menu principal <Ctrl-M>, de passer à l'écran suivant <Ctrl-E> et de retourner au début de l'écran en cours <Ctrl-D>. Il est impossible de retourner directement à l'écran précédent. Pour le faire, il faut retourner au menu principal <Ctrl-M> et faire défiler les écrans <Ctrl-E> jusqu'à l'apparition de celui désiré. Les données sont regroupées en catégories contenant chacune un ou plusieurs écrans. On a:

- type de calcul et configuration globale (1 écran),
- configurations des donnees (1 écran),
- données générales (2 ou 3 écrans),
- données spécifiques (1 à 4 écrans), et
- restrictions et paramètres de contrôle (1 écran).

5.3 CALCUL

Cette option met en marche le calcul d'un circuit selon les spécifications fournies. On peut suivre à l'écran l'évolution du calcul. Le temps de calcul varie selon le type d'ordinateur, la dimension du circuit et la précision voulue. Pour donner un ordre de grandeur, un PC-XT prend généralement entre 1 et 3 minutes pour calculer un circuit.

5.4 IMPRIMER

Cette option permet une impression plus ou moins détaillée selon le format spécifié. On peut imprimer les données d'entrée, les résultats concernant l'équipement et ceux donnant le bilan. Le format du bilan sert à définir le type de bilan à imprimer. On peut obtenir un bilan simplifié, détaillé ou sur graphique. Le graphique peut être un schéma du circuit complet ou des courbes Distribution-Teneur en fonction du nombre de cellules. Le schéma ne peut contenir qu'une information soit: la teneur, la distribution ou le débit d'un constituant. On peut visualiser 2 courbes simultanément. Pour chaque courbe on doit définir la nature de l'ordonnée (cumulée ou non), le circuit ainsi que le constituant. On présente en Annexe 2 des exemples d'impressions.

5.5 LIRE - SAUVEGARDER - EFFACER

Ces trois options contrôlent la gestion des fichiers. Ces fichiers APL renferment les données d'entrée ainsi que l'ensemble des résultats. On ne peut pas avoir accès au contenu des fichiers avec le DOS. Les fichiers formés ont le suffixe .ASF et commencent tous par FL .

5.6 TERMINER

Cette option permet de terminer l'exécution de CALFLOT et de retourner au système d'exploitation (DOS).

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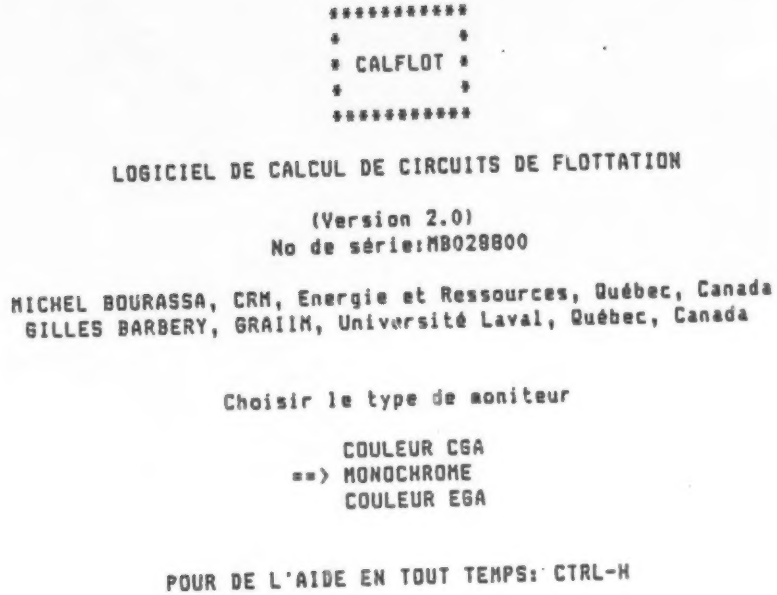


```
                    ***********
                    *         *
                    * CALFLOT *
                    *         *
                    ***********

      LOGICIEL DE CALCUL DE CIRCUITS DE FLOTTATION

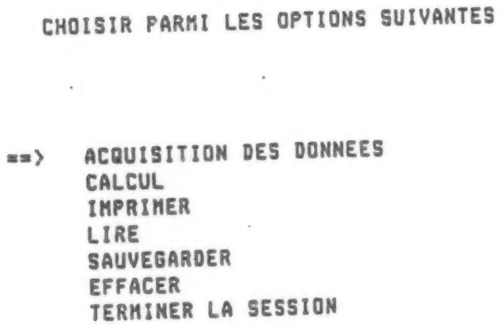
                     (Version 2.0)
                 No de série:MB028800

  MICHEL BOURASSA, CRM, Energie et Ressources, Québec, Canada
   GILLES BARBERY, GRAIIM, Université Laval, Québec, Canada


               Choisir le type de moniteur

                    COULEUR CGA
                ==> MONOCHROME
                    COULEUR EGA


        POUR DE L'AIDE EN TOUT TEMPS: CTRL-H
```



```
     CHOISIR PARMI LES OPTIONS SUIVANTES



==>   ACQUISITION DES DONNEES
      CALCUL
      IMPRIMER
      LIRE
      SAUVEGARDER
      EFFACER
      TERMINER LA SESSION
```

```
                    TYPE DE CALCUL ET CONFIGURATION GLOBALE


=====>  CHOISIR LE TYPE DE CALCUL

                             ébaucheur-épuiseur (banc)
                         ==> ébaucheur-épuiseur et nettoyeur (complet)

=====>  CHOISIR LA CONFIGURATION GLOBALE

     EBAUCHEURS              avec épuiseurs
                             sans épuiseurs
                         ==> au choix du programme

     1er NETTOYEURS          avec épuiseurs et retour aux ébaucheurs
                             avec épuiseurs et retour aux 1er nettoyeurs
                             sans épuiseur et retour aux ébaucheurs
                             sans épuiseur et pas de retour
                         ==> au choix du programme

     AUTRES NET.             avec 2ème nettoyeurs
                             avec 2ème et 3ème nettoyeurs
                         ==>.au choix du programme
```

```
                         CONFIGURATION DES DONNEES


                                                   EBAU  NET1  NET2  NET3
Paramètres cinétiques connus (R.inf.,Kf,Ks,Phi) ?   N     N     N     N
Coefficients d'entrainement connus (CF) ?           N     -     -     -
Constante cinétique de l'eau connue?                N     N     N     N
Volume des cellules imposé ?                        N     N     N     N
Marque des cellules imposée ?                       N     N     N     N
Nombre de cellules par banc imposé?                 N     N     N     N
Nombre de bancs imposé?                             N     N     N     N
Débit d'eau pour écraser l'écume connu ?            N     N     -     -
Paramètre Phi des mixtes connu ?                    N     N     -     -
Nombre de cellules aux épuiseurs fixé ?             N     N     N     N
Teneur minimale du concentré imposée ?

                         - : Non applicable
                         DONNEES GENERALES (1)

                                                              100
Tonnage horaire de solide (T.M./Hre)                          1
Nb de classes granulométriques (Max 5)                        1
Nb de minéraux hydrophobes ou flottés (Max 2)                 2
Nb de minéraux hydrophiles (Max 5)

              Nom du minéral hydrophobe 1           CHALCOPYR

              Nom du minéral hydrophile 1           Calcite
              Nom du minéral hydrophile 2           Pyrite

Teneur minimale en CHALCOPYR du produit final (frac.) ?     0.85
```

```
                         DONNEES GENERALES (2)


    MASSE VOLUMIQUE RELATIVE

      CHALCOPYR    4.2
      Calcite      2.7
      Pyrite       5

TENEUR MINERALE A L'ALIMENTATION (f.)

    CHALCOPYR          .20
    Calcite            .65
    Pyrite             .15
                      ------
                       1.00
```

```
                         DONNEES EBAUCHEUR

                      CHALCOPY                 Minéraux types
Choisir le minéral qui                      Barytine
s'apparente le plus                         Charbon
au(x) vôtre(s)            ==>               Chalcopyrite
                                            Chalcopyrite flottation lente
                                            Fluorite
                                            Nickel
                                            Galène
                                            Galène flottation lente
                                            Sphalérite (blende)
                                            Wolframite-scheelite

Conc. solides (mas.) à l'Alim. (f.)          .3500
```

```
                         DONNEES NETTOYEUR 1

                      CHALCOPY                 Minéraux types
Choisir le minéral qui                      Barytine
s'apparente le plus                         Charbon
au(x) vôtre(s)            ==>               Chalcopyrite
                                            Chalcopyrite flottation lente
                                            Fluorite
                                            Nickel
                                            Galène
                                            Galène flottation lente
                                            Sphalérite (blende)
                                            Wolframite-scheelite

Conc. solides (mas.) à l'Alim. (f.)          .3000
```

```
                          DONNEES NETTOYEUR 2

                     CHALCOPY                         Minéraux types
                                                    Barytine
Choisir le minéral qui                              Charbon
s'apparente le plus                                 Chalcopyrite
au(x) vôtre(s)              ==>                     Chalcopyrite flottation lente
                                                    Fluorite
                                                    Nickel
                                                    Galène
                                                    Galène flottation lente
                                                    Sphalérite (blende)
                                                    Wolframite-scheelite

Conc. solides (mas.) à l'Alim. (f.)             .3000
```

```
                          DONNEES NETTOYEUR 3

                     CHALCOPY                         Minéraux types
                                                    Barytine
Choisir le minéral qui                              Charbon
s'apparente le plus                                 Chalcopyrite
au(x) vôtre(s)              ==>                     Chalcopyrite flottation lente
                                                    Fluorite
                                                    Nickel
                                                    Galène
                                                    Galène flottation lente
                                                    Sphalérite (blende)
                                                    Wolframite-scheelite

Conc. solides (mas.) à l'Alim. (f.)             .2800
```

```
                 RESTRICTIONS ET PARAMETRES DE CONVERGENCE

                               RESTRICTIONS

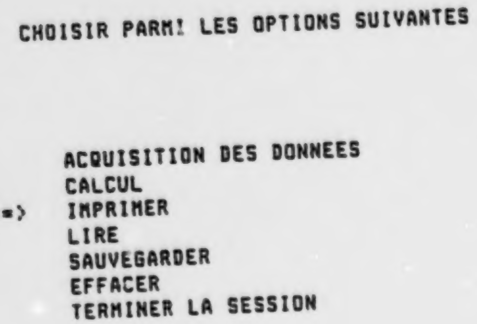
        CHOISIR LES VALEURS DESIREES       Valeurs en mémoire
                                       ==) Valeurs recommandées

                                                EBAUCH   NET1    NET2    NET3
Rendement max. pour une cellule (f.)            .3000   .4000   .4000   .4000
Rendement min. justifiée pour une cel. (f.)     .0050   .0050   .0050   .0050
Conc. solides max. pour un concentré (f.)       .3500   .4000   .4000   .4000

                         PARAMETRES DE CONVERGENCE

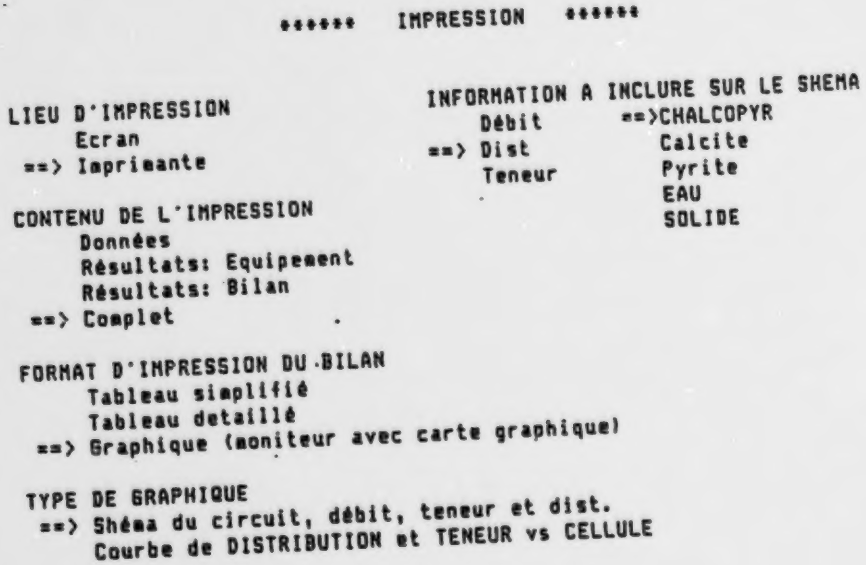
        CHOISIR LES VALEURS DESIREES       Valeurs en mémoire
                                       ==) Valeurs recommandées
             Seuil de conv.: temps de séjour (f.)       .0100
             Seuil de conv.: gangue entrainée (f.)      .0100
             Seuil de conv.: flux recirculés (f.)       .0100
             Nb d'itérations pour le temps de séjour        1
             Nb d'itérations pour la gangue entrainée       8
             Nb d'itérations pour les flux recirculés      10
```

ANNEXE B



```
CHOISIR PARMI LES OPTIONS SUIVANTES


      ACQUISITION DES DONNEES
      CALCUL
==>   IMPRIMER
      LIRE
      SAUVEGARDER
      EFFACER
      TERMINER LA SESSION
```



```
                    ******   IMPRESSION   ******

                              INFORMATION A INCLURE SUR LE SHEMA
LIEU D'IMPRESSION                 Débit      ==>CHALCOPYR
     Ecran                    ==> Dist          Calcite
 ==> Imprimante                   Teneur        Pyrite
                                                EAU
CONTENU DE L'IMPRESSION                         SOLIDE
     Données
     Résultats: Equipement
     Résultats: Bilan
 ==> Complet

FORMAT D'IMPRESSION DU BILAN
     Tableau simplifié
     Tableau detaillé
 ==> Graphique (moniteur avec carte graphique)

TYPE DE GRAPHIQUE
 ==> Shéma du circuit, débit, teneur et dist.
     Courbe de DISTRIBUTION et TENEUR vs CELLULE
```

```
****** IMPRESSION : DONNEES D'ENTREE ******


TYPE DE CALCUL:  Ebaucheur-epuiseur et nettoyeur (complet)

CONFIGURATION GLOBALE,
                 EBAUCHEURS:      au choix du programme
                 1er NETTOYEURS:  au choix du programme
                 AUTRES NETTOY.:  au choix du programme

                                                          EB  N1  N2
  CONFIGURATION DES CIRCUITS:
Paramètres cinétiques connus (R.inf.,Kf,Ks,Phi) ?         N   N   N
Coefficients d'entrainement connus (CF) ?                 N   -   -
Constante cinétique de l'eau connue?                      N   N   N
Volume des cellules imposé ?                              N   N   N
Marque des cellules imposée ?                             N   N   N
Nombre de cellules par banc imposé?                       N   N   N
Nombre de bancs imposé?                                   N   N   N
Débit d'eau pour écraser l'écume connu ?                  N   N   N
Paramètre Phi des mixtes connu ?                          N   N   -
Nombre de cellules aux épuiseurs fixé ?                   N   N   -
Teneur minimale du concentré imposée ?                    N   N   N


Tonnage horaire de solides (T.M./h)=    100
Teneur min. en CHALCOPYR du produit final (%)=  85

        Masse Volumique Relative
          CHALCOPYR    4.2
          Calcite      2.7
          Pyrite       5


                  TENEURS MINERALES A L'ALIMENTATION (frac.)

      CHALCOPYR         .2000
      Calcite           .6500
      Pyrite            .1500
                        ------
                        1.000


                        Coefficient d'entrainement


                           Calcite      .70
                           Pyrite       .70
```

	EBAUC CHALCO	NET 1 CHALCO	NET 2 CHALCO	NET 3 CHALCO
Kf	.729	1.184	1.184	1.184
Fact. echelle sur Kf	1.000	1.000	1.000	1.000
Ks	.073	.118	.118	.118
Fact. échelle sur Ks	1.000	1.000	1.000	1.000
Phi	.200	.200	.200	.200
Phi des mixtes	.200	.200	.200	.200
R. infini	.950	1.000	1.000	1.000

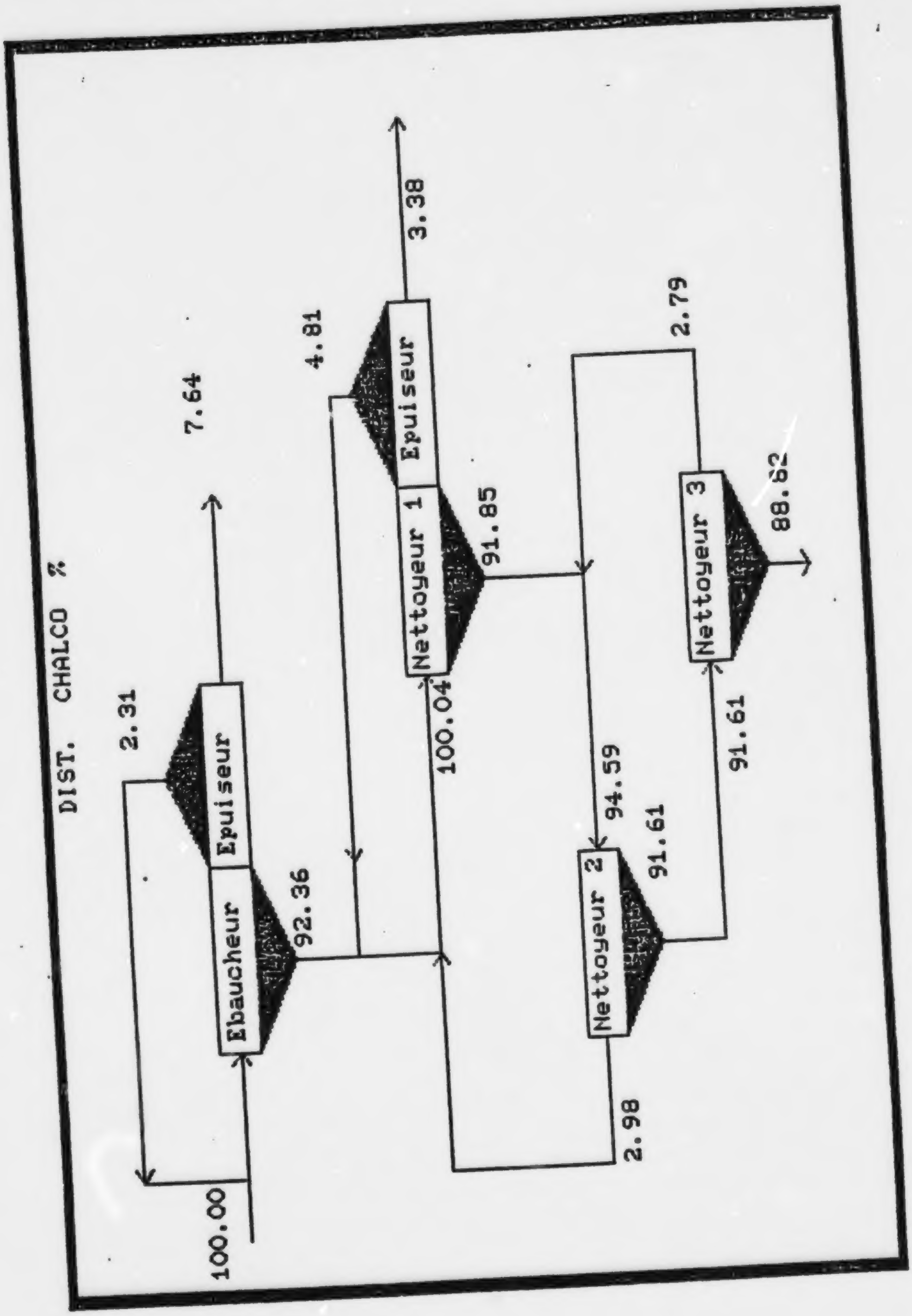
	EBAUCH	NET1	NET2	NET3
Conc. solides (mas.) à l'Alim. (f.)	.350	.300	.300	.280
Constante cinétique de l'eau (1/min.)	.005	.008	.010	.010

	EBAUCH	NET1	NET2	NET3
Rendement max. pour une cellule (f.)	.300	.400	.400	.400
Rendement min. justifiée pour une cel. (f.)	.0050	.0050	.0050	.0050
Conc. solides max. pour un concentré (f.)	.350	.400	.400	.400
Seuil de conv.: temps de séjour (f.)	.0100			
Seuil de conv.: gangue entrainée (f.)	.0100			
Seuil de conv.: flux recirculés (f.)	.0100			
Nb d'itérations pour le temps de séjour	1			
Nb d'itérations pour la gangue entrainée	8			
Nb d'itérations pour les flux recirculés	10			

```
****** IMPRESSION - RESULTATS - EQUIPEMENTS ******

   CIRCUIT                         Ebauch Epuise  Net1  Ep-Net1  Net2   Net3
Nombre de bancs:                     1      1      1      1       1      1
Nombre de cellules/banc:             8      3      5      4       8      6
Vol. des cellules (m3):            8.50   8.50   4.25   4.25    1.78   1.78
                  (pi3):          300.0  300.0  150.0  150.0    63.0   63.0
Type de cellules:                 simple simple simple simple double double
Puissance installée (HP):          30.0   30.0   15.0   15.0    20.0   20.0
Manufacturier(s) * :               WAD    WAD     W      W       A      A
Temps de flot. (min):              20.6    9.5    8.1    7.7    14.1   12.8
Temps de flot. unitaire:           2.58   3.16   1.62   1.92    1.76   2.13
Volume possible (m3):              5.66   5.66   2.83   2.83    1.13   1.13
                                   7.99   7.99   3.00   3.00    1.50   1.50
                                  14.16  14.16                  1.73   1.73


*  MANUFACTURIER: (W)Wemco, (A)Agitair, (O) Outokumpu, (D)Dorr
```


DIST. CHALCO %
100.00
2.31
7.64
Ebaucheur
Epuiseur
92.36
4.81
3.38
Nettoyeur 1
Epuiseur
91.85
100.04
94.59
Nettoyeur 2
91.61
2.98
Nettoyeur 3
2.79
88.82
91.61

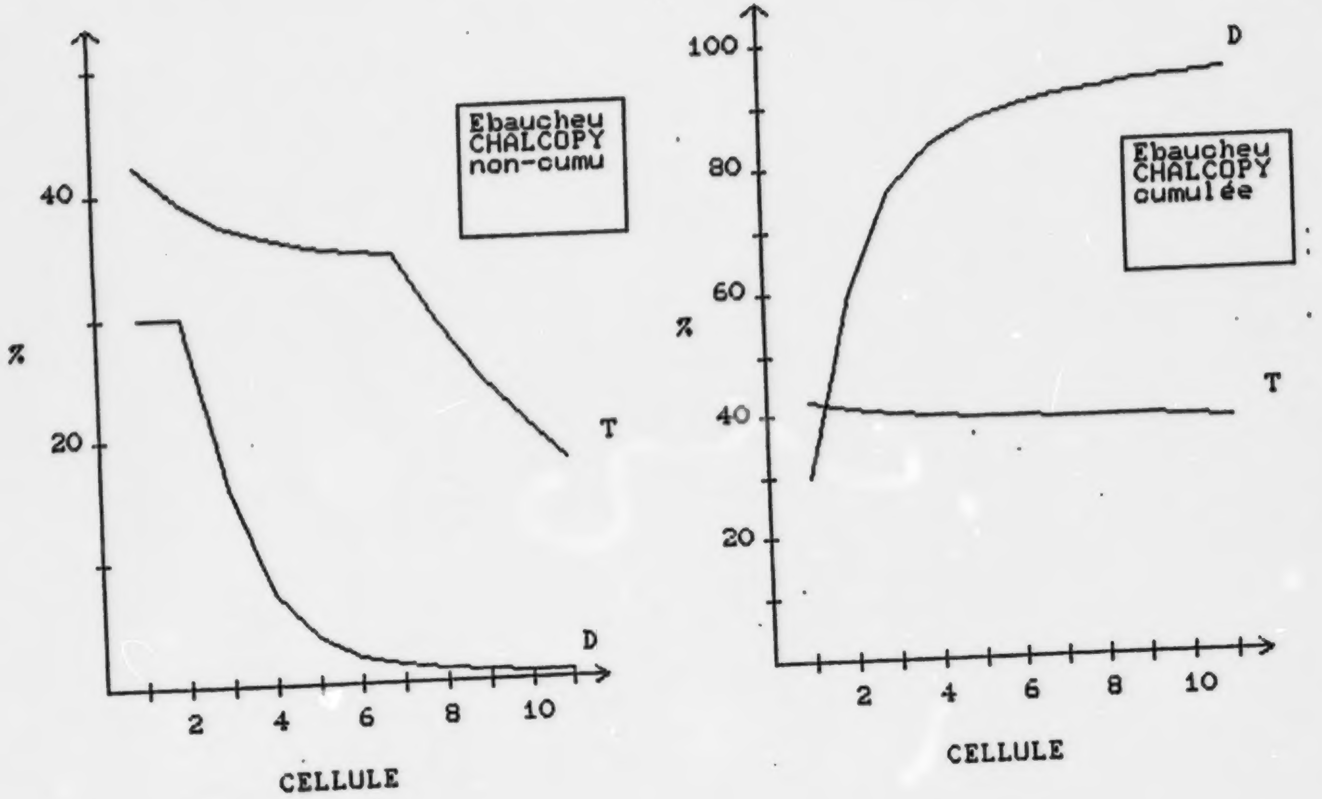
```
                    ******   IMPRESSION   ******

LIEU D'IMPRESSION               TYPE DE COURBE DISTRIBUTION-TENEUR
     Ecran                      ==> Ebaucheur  ==>CHALCOPYR
 ==> Imprimante                     Nettoy 1      Calcite
                                    Nettoy 2      Pyrite
CONTENU DE L'IMPRESSION             Nettoy 3      EAU
     Données                                      SOLIDE
     Résultats: Equipement
 ==> Résultats: Bilan
     Complet

FORMAT D'IMPRESSION DU BILAN
     Tableau simplifié
     Tableau detaillé
 ==> Graphique (moniteur avec carte graphique)

TYPE DE GRAPHIQUE
     Shéma du circuit, débit, teneur et dist.
 ==> Courbe de DISTRIBUTION et TENEUR vs CELLULE
COURBE 2     ==> non-cumulée
             ==> cumulée
```

Distribution (D) et Teneur (T) VS Cellule



APPENDIX 7

Rate Parameter Test for Antimony Flotation with Varying Test Procedures

PROCESS STUDIES GROUP	technical notes	Project No.	File No.
	Project Flotation Simulation	Date 18/08/88	Page 1

Subject: Rate Parameter Tests for Antimony Flotation with Varying Test Procedures

Author(s): Donald S. Taylor

Objective: To determine if the varying of flotation test procedures would affect the flotation kinetic rate constant for a given ore, and in turn the circuit design as calculated by the flotation simulator CALFLOT.

Conclusions: 1. The flotation test procedures affect the kinetic rate data:

TEST	KINETIC RATE PARAMETERS			
	Kf	Ks	O	R
Antimony I	0.585	0.025	0.122	0.883
Antimony II	0.806	0.073	0.215	0.759
Antimony III	0.692	0.033	0.205	0.802

2. The kinetic rate parameters affect the circuit designed by the flotation simulator CALFLOT:

TEST	FLOTATION SIMULATION RESULTS			
	Mineral Grade	Antimony Grade	Recovery	Cells
Antimony I	69.0	49.47	81.1	8
Antimony II	61.4	44.02	72.7	11
Antimony III	58.1	41.66	[illegible]	[illegible]

Author(s) Donald S. Taylor	*technical notes*	*Project No.*	*Page* 2 *of* 2

Discussion: Tests were completed on antimony ore from Dominion Explorers to determine if the laboratory test procedures would affect kinetic rate data and CALFLOT design parameters. The tests were done on the same ore but the reagent addition quantities and placement of the reagent additions in the test sequence were varied. The best results were obtained from a single reagent addition at the beginning of the test. Figures 1 to 3 show the test data including reagent addition quantities and addition points.

From the data, it is clear that the laboratory procedures affect the rates of kinetic reactions and therefore circuit design. Thus, before kinetic reaction rate tests are run, the best set of laboratory test procedures need to be obtained for the ore that is being tested. Then proper rate reactions can be obtained for circuit design.

Method: The tests were carried out as shown in Figures 1 to 3, then the resulting flotation concentrates were dried, weighed and assayed. The data was then placed on a standard metallurgical balance computer program with the results shown in Figures 4 to 6. This data was then put on CALFLOT, the kinetic rate calculator program, and we obtained, at 1% accuracy, the rates shown in the table on the previous page, and shown in Figures 7 to 9. The flotation rate data was run in the CALFLOT program, and the results are shown in the table on the previous page and in Attachments 1 to 3.

FIGURE 1

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Antimony Feed
SAMPLE WEIGHT: 1,000 g
PARTICLE SIZE: 1 hr grind
PURPOSE: Rate Parameter Tests on Antimony Ore
BACKGROUND:

PROJECT NUMBER:
TEST NUMBER: Antimony I
TEST BY: B. Blizzard
DATE: May 22, 1988
CLIENT: Dominion Explorers

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)						
					Aero Prom 3477 10%	MIBC	Pb Nitrate				
Reagents			Natural		0.25 g	0.03 g	0.62 g				
Float 1	0.50										
Float 2	0.50										
Float 3	1.0										
Float 4	2.0										
Float 5	4.0										
Float 6	8.0										
Float 7	8.0										

COMMENTS: All reagents added at start of test

FIGURE 2

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE:	Antimony Feed	**PROJECT NUMBER:**	
SAMPLE WEIGHT:	1,000 g	**TEST NUMBER:**	Antimony III
PARTICLE SIZE:	1 hr grind	**TEST BY:**	B. Blizzard
PURPOSE:	Rate Parameter Tests on Antimony Ore	**DATE:**	June 3, 1988
BACKGROUND:		**CLIENT:**	Dominion Explorers

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)						
					Aero Prom 3477 10%	MIBC	Pb Nitrate				
Reagents			Natural		0.26 g	0.03 g	0.62 g				
Float 1	0.50										
Float 2	0.50										
Float 3	1.0										
Float 4	2.0										
Float 5	4.0										
Reagents					0.13 g	0.02 g	0.31 g				
Float 6	8.0										
Float 7	8.0										

COMMENTS: Normal reagent dosage at start of test, then at 8 minutes total flotation time more reagents addded.

FIGURE 3

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Antimony Feed
SAMPLE WEIGHT: 1,000 g
PARTICLE SIZE: 1 hr grind
PURPOSE: Rate Parameter Tests on Antimony Ore
BACKGROUND:

PROJECT NUMBER:
TEST NUMBER: Antimony II
TEST BY: B. Blizzard
DATE: May 26, 1988
CLIENT: Dominion Explorers

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)						
					Aero Drum 3477 10%	MIBC	Pb Nitrate				
Reagents			Natural		0.13 g	0.03 g	0.62 g				
Float 1	0.50										
Float 2	1.5										
Float 3	2.0										
Float 4	4.0										
Reagents					0.13 g	0.015 g	0.31 g				
Float 5	8.0										
Float 6	8.0										

COMMENTS: Reagent dosage split one-half at start one-half after 8 minutes of total flotation time.

FIGURE 4

GRADE AND RECOVERY

METAL: Sb SAMPLE NAME ANIMONY FLOAT I FILE NAME CALFLOT

SAMPLE	WEIGHT (grams)	WEIGHT PERCENT	METAL ASSAY	METAL CONTENT	METAL DISTRIBUTION	GRADE	CUMULATIVE % WEIGHT	CUMULATIVE % RECOVERY
TAILS	1126.84	93.87	0.62	58.20	17.63	44.31	6.13	82.37
24	2.18	0.18	20.70	3.76	1.14	0.00	5.95	81.23
16	3.81	0.32	27.40	8.70	2.63	0.00	5.64	78.59
8	8.96	0.75	34.80	25.97	7.87	0.00	4.89	70.72
4	14.78	1.23	40.90	50.35	15.26	0.00	3.66	55.47
2	17.63	1.47	48.40	71.08	21.54	0.00	2.19	33.93
1	12.72	1.06	46.80	49.59	15.02	0.00	1.13	18.91
.30	13.57	1.13	55.20	62.40	18.91	0.00	0.00	0.00
	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
FEED	1200.49		3.30	330.04				

BALANCE: 100.01

FIGURE 5

GRADE AND RECOVERY

SAMPLE NAME ANIMONY FLOAT II FILE NAME CALFLOT

METAL: Sb

SAMPLE	WEIGHT (grams)	WEIGHT PERCENT	METAL ASSAY	METAL CONTENT	METAL DISTRIBUTION	GRADE	CUMULATIVE % WEIGHT	CUMULATIVE % RECOVERY
TAILS	929.82	93.92	1.00	93.92	27.07	41.62	6.08	72.93
24	5.45	0.55	10.40	5.73	1.65	0.00	5.53	71.27
16	10.20	1.03	15.50	15.97	4.60	0.00	4.50	66.67
8	7.94	0.80	28.10	22.54	6.50	0.00	3.70	60.17
4	6.24	0.63	45.50	28.68	8.27	0.00	3.07	51.91
2	19.87	2.01	57.70	115.81	33.38	0.00	1.06	18.52
.30	10.48	1.06	60.70	64.26	18.52	0.00	-0.00	0.00
N/A	0.00	0.00	0.00	0.00	0.00	0.00	-0.00	0.00
	0.00	0.00	0.00	0.00	0.00	0.00	-0.00	0.00
	0.00	0.00	0.00	0.00	0.00	0.00	-0.00	0.00
	0.00	0.00	0.00	0.00	0.00	0.00	-0.00	0.00
FEED	990.00		3.30	346.90				

BALANCE: 105.12

FIGURE 6

GRADE AND RECOVERY

METAL: Sb SAMPLE NAME ANIMONY FLOAT III FILE NAME CALFLOT

SAMPLE	WEIGHT (grams)	WEIGHT PERCENT	METAL ASSAY	METAL CONTENT	METAL DISTRIBUTION	GRADE	CUMULATIVE x WEIGHT	CUMULATIVE x RECOVERY
TAILS	990.14	94.15	0.95	89.44	27.42	40.45	5.85	72.58
24	5.42	0.52	9.10	4.69	1.44	0.00	5.34	71.14
16	9.46	0.90	16.70	15.02	4.61	0.00	4.44	66.53
8	5.45	0.52	29.80	15.44	4.73	0.00	3.92	61.80
4	11.34	1.08	36.60	39.46	12.10	0.00	2.84	49.70
2	10.59	1.01	53.50	53.87	16.52	0.00	1.83	33.18
1	9.68	0.92	58.70	54.03	16.57	0.00	0.91	16.61
.30	9.61	0.91	59.30	54.19	16.61	0.00	0.00	0.00
	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
FEED	1051.69		3.30	326.15				

BALANCE: 98.83

FIGURE 7

```
Test Identification : ANTIMONY 1

  Cumulative Time          Cumulative Recovery
       (min)            measured          calculated
1      0.5               0.1891              .1981
2      1                 0.3393              .3462
3      2                 0.5547              .5400
4      4                 0.7072              .7109
5      8                 0.7859              .7881
6      16                0.8123              .8114
7      24                0.8237              .8231
8
9
10

Parameter     Start Value     Relative          Optimal Value
                            accuracy (%)
  Kf             0.8             5                  .587
  Ks             0.1             5                  .043
  Phi            0.2             5                  .093
 R inf.          0.86            5                  .851

       ROOT SQARE OF RESIDUES : 0.019083
```

```
Test Identification : ANTIMONY 1

  Cumulative Time          Cumulative Recovery
       (min)            measured          calculated
1      0.5               0.1891              .1981
2      1                 0.3393              .3462
3      2                 0.5547              .5402
4      4                 0.7072              .7112
5      8                 0.7859              .7878
6      16                0.8123              .8108
7      24                0.8237              .8239
8
9
10

Parameter     Start Value     Relative          Optimal Value
                            accuracy (%)
  Kf             0.8             1                  .585
  Ks             0.1             1                  .025
  Phi            0.2             1                  .122
 R inf.          0.86            1                  .883

       ROOT SQARE OF RESIDUES : 0.018988
```

FIGURE 8

```
Test Identification : ANTIMONY 2

  Cumulative Time           Cumulative Recovery
       (min)             measured          calculated
1       0.5              0.1852               .2025
2       2                0.5191               .4989
3       4                0.6017               .6142
4       8                0.6667               .6673
5       16               0.7127               .7079
6       24               0.7293               .7313
7
8
9
10

Parameter      Start Value      Relative           Optimal Value
                               accuracy (%)
  Kf              1                5                   .795
  Ks              0.1              5                   .066
  Phi             0.2              5                   .214
 R inf.           0.75             5                   .765

        ROOT SQARE OF RESIDUES : 0.029879
```

```
Test Identification : ANTIMONY 2

  Cumulative Time           Cumulative Recovery
       (min)             measured          calculated
1       0.5              0.1852               .2034
2       2                0.5191               .4991
3       4                0.6017               .6136
4       8                0.6667               .6674
5       16               0.7127               .7087
6       24               0.7293               .7311
7
8
9
10

Parameter      Start Value      Relative           Optimal Value
                               accuracy (%)
  Kf              1                1                   .806
  Ks              0.1              1                   .073
  Phi             0.2              1                   .215
 R inf.           0.75             1                   .759

        ROOT SQARE OF RESIDUES : 0.029864
```

FIGURE 9

```
Test Identification : ANTIMONY 3

  Cumulative Time          Cumulative Recovery
        (min)            measured         calculated
1      0.5                0.1661            .1892
2      1                  0.3318            .3238
3      2                  0.4970            .4882
4      4                  0.6180            .6176
5      8                  0.6653            .6734
6      16                 0.7114            .7057
7      24                 0.7258            .7276
8
9
10

Parameter      Start Value      Relative          Optimal Value
                               accuracy (%)
  Kf             .7               5                     .694
  Ks             0.1              5                     .038
  Phi            0.2              5                     .195
 R inf.          0.76             5                     .789

       ROOT SQARE OF RESIDUES : 0.027922
```

```
Test Identification : ANTIMONY 3

  Cumulative Time          Cumulative Recovery
        (min)            measured         calculated
1      0.5                0.1661            .1891
2      1                  0.3318            .3237
3      2                  0.4970            .4883
4      4                  0.6180            .6180
5      8                  0.6653            .6734
6      16                 0.7114            .7052
7      24                 0.7258            .7277
8
9
10

Parameter      Start Value      Relative          Optimal Value
                               accuracy (%)
  Kf             .7               1                     .692
  Ks             0.1              1                     .033
  Phi            0.2              1                     .205
 R inf.          0.76             1                     .802

       ROOT SQARE OF RESIDUES : 0.027904
```

```
                                   ANTIMONY I

                           ****** PRINT : INPUT DATA ******

     CALCULATION MODE: Rougher-Scavenger Alone

     CIRCUIT CONFIGURATION
 ROUGHER: Chose by Programm

     DATA CONFIGURATION:                                          RS
    Kinetic Parametres Known (R.inf.,Kf,Ks,Phi) ?                  Y
    Entrainment Coef. for Unfloat minerals Known (CF) ?            Y
    Rate Constants for Water known ?                               N
    Volume Cells fixed ?                                           N
    Mark of Cells fixed ?                                          N
    Number of Cells per Bank fixed ?                               N
    Number of Cells Fixed ?                                        N
    Water Flowrate for Froth Collapse known ?                      N
    Proportion of Slow Floating in mids known ?                    N
    Number of Cells for Scavenger fixed ?                          N
    Minimum Grade for Concentrate fixed ?                          N




   Solid Flowrate (T.M./h)=                     100


           Relative Density
         ANTIMONY     4.7
         QUARTZ       2.6


                            MINERAL FEED GRADE (frac.)

       ANTIMONY          .0466
       QUARTZ            .9534
                         ------
                         1.000


                           Entrainment Coefficients (CF)


                                QUARTZ       .16
```

```
                          RGHR
                          ANTIMO

Kf:Fast Rate(1/min)        .585
Scale-up Fact on Kf       1.000
Ks:Slow Rate(1/min)        .025
Scale-up Fact on Ks       1.000
Slow prop. (f)             .122
Slow Prop in mids          .122
Ultimate Rec. (f)          .883
```

```
                                             RGHR
Percent of Solids in Feed (f.)               .420
Water rate constant (l/min.)                 .005
```

```
                                                    RGHR
Maximum Recovery for Cells (f)                      .300
Minimum Recovery Justified for a Cell (f)          .0050
Max Percent Solid for Concentrates (f)              .350
Tolerance for Residence Time Calcul. (f)           .0100
Tolerance for Entrainment Calculation (f)          .0100
Tolerance for Circulating Flow Calcul. (f)         .0100
Max Iterations for Residence Time Calcul.              1
Max Iterations for Entrainment Calculation             8
Max Iterations for Circulating Flow Calcul.           10
```

```
                    ****** PRINT : EQUIPEMENT ******


      CIRCUIT                      Rghr
Number of Parnl. Banks:              1
Number of Cells/Bank  :              8
Volume of Cells (m3)  :            7.99
                (pi3) :            282.0
Type of Cells:                     simple
Power Installed (HP):              20.0
Manufacturer(s) * :                  O
Flotation Time (min):              17.2
Unit Flotation Time :              2.15
Alternative Volume(m3):            4.25
                                   5.66
                                   8.50


   *  MANUFACTURER: (W)Wemco, (A)Agitair, (O) Outokumpu, (D)Dorr
```

```
                    ****** RESULTS : SUMMARY BALANCE ******

                              CIRCUIT : ROUGHER

                                      FLOWRATE t/h
                         New Feed        Rougher Conc         Tail
Solid                    100.000            5.471            94.529
Water                    138.095           14.542           123.553
Water Addition                              2.425
Water adjusment                              .000

                                 GRADE  and DISTRIBUTION %
                          New Feed       Rougher Conc           Tail
                      Grade     Dist     Grade     Dist     Grade      Dist
ANTIMONY               4.7     100.0      69.0     81.1        .9      18.9
QUARTZ                95.3     100.0      31.0      1.8      99.1      98.2
Water                 58.0     100.0      72.7     10.5      56.7      89.5
Solid                 42.0     100.0      27.3      5.5      43.3      94.5
```

ATTACHMENT 2

ANTIMONY 2

```
                    ****** PRINT : INPUT DATA ******


    CALCULATION MODE: Rougher-Scavenger Alone

    CIRCUIT CONFIGURATION
ROUGHER: Chose by Programm

    DATA CONFIGURATION:                                           RS
  Kinetic Parametres Known (R.inf.,Kf,Ks,Phi) ?                  Y
  Entrainment Coef. for Unfloat minerals Known (CF) ?            Y
  Rate Constants for Water known ?                               N
  Volume Cells fixed ?                                           N
  Mark of Cells fixed ?                                          N
  Number of Cells per Bank fixed ?                               N
  Number of Cells Fixed ?                                        N
  Water Flowrate for Froth Collapse known ?                      N
  Proportion of Slow Floating in mids known ?                    N
  Number of Cells for Scavenger fixed ?                          N
  Minimum Grade for Concentrate fixed ?                          N



  Solid Flowrate (T.M./h)=                      100


          Relative Density
        ANTIMONY     4.7
        QUARTZ       2.6


                         MINERAL FEED GRADE (frac.)

     ANTIMONY          .0466
     QUARTZ            .9534
                       ------
                       1.000


                        Entrainment Coefficients (CF)


                          QUARTZ        .16
```

	RGHR ANTIMO
Kf:Fast Rate(1/min)	.806
Scale-up Fact on Kf	1.000
Ks:Slow Rate(1/min)	.073
Scale-up Fact on Ks	1.000
Slow prop. (f)	.215
Slow Prop in mids	.215
Ultimate Rec. (f)	.759

	RGHR
Percent of Solids in Feed (f.)	.420
Water rate constant (1/min.)	.005

	RGHR
Maximum Recovery for Cells (f)	.300
Minimum Recovery Justified for a Cell (f)	.0050
Max Percent Solid for Concentrates (f)	.350
Tolerance for Residence Time Calcul. (f)	.0100
Tolerance for Entrainment Calculation (f)	.0100
Tolerance for Circulating Flow Calcul. (f)	.0100
Max Iterations for Residence Time Calcul.	1
Max Iterations for Entrainment Calculation	8
Max Iterations for Circulating Flow Calcul.	10

```
                    ****** PRINT : EQUIPEMENT ******

     CIRCUIT                        Rghr
Number of Paral. Banks:               1
Number of Cells/Bank  :              11
Volume of Cells (m3)  :             7.99
                (pi3) :             282.0
Type of Cells:                      simple
Power Installed (HP):               20.0
Manufacturer(s) * :                  O
Flotation Time (min):               23.9
Unit Flotation Time :               2.18
Alternative Volume(m3):             4.25
                                    5.66
                                    8.50


   *  MANUFACTURER: (W)Wemco, (A)Agitair, (O) Outokumpu, (D)Dorr
```

```
                ****** RESULTS : SUMMARY BALANCE ******

                          CIRCUIT : ROUGHER

                                    FLOWRATE t/h
                      New Feed     Rougher Conc       Tail
Solid                 100.000         5.517          94.483
Water                 138.095        18.089         120.006
Water Addition                        2.567
Water adjusment                        .000

                            GRADE  and DISTRIBUTION %
                      New Feed       Rougher Conc         Tail
                   Grade    Dist    Grade    Dist    Grade    Dist
ANTIMONY             4.7   100.0     61.4    72.7      1.3    27.3
QUARTZ              95.3   100.0     38.6     2.2     98.7    97.8
Water               58.0   100.0     76.6    13.1     55.9    86.9
Solid               42.0   100.0     23.4     5.5     44.1    94.5
```

ATTACHMENT 3

ANTIMONY 3

```
                    ****** PRINT : INPUT DATA ******

   CALCULATION MODE: Rougher-Scavenger Alone

   CIRCUIT CONFIGURATION
ROUGHER: Chose by Programm

   DATA CONFIGURATION:                                          RS
 Kinetic Parametres Known (R.inf.,Kf,Ks,Phi) ?                   Y
 Entrainment Coef. for Unfloat minerals Known (CF) ?             Y
 Rate Constants for Water known ?                                N
 Volume Cells fixed ?                                            N
 Mark of Cells fixed ?                                           N
 Number of Cells per Bank fixed ?                                N
 Number of Cells Fixed ?                                         N
 Water Flowrate for Froth Collapse known ?                       N
 Proportion of Slow Floating in mids known ?                     N
 Number of Cells for Scavenger fixed ?                           N
 Minimum Grade for Concentrate fixed ?                           N



 Solid Flowrate (T.M./h)=                       100

          Relative Density
        ANTIMONY     4.7
        QUARTZ       2.6


                          MINERAL FEED GRADE (frac.)

   ANTIMONY          .0466
   QUARTZ            .9534
                     ------
                     1.000


                          Entrainment Coefficients (CF)


                              QUARTZ        .16
```

```
                         RGHR
                         ANTIMO

Kf:Fast Rate(1/min)       .692
Scale-up Fact on Kf      1.000
Ks:Slow Rate(1/min)       .033
Scale-up Fact on Ks      1.000
Slow prop. (f)            .205
Slow Prop in mids         .205
Ultimate Rec. (f)         .802
```

```
                                          RGHR
                                          .420
Percent of Solids in Feed (f.)            .005
Water rate constant (l/min.)
```

```
                                                  RGHR
                                                  .300
Maximum Recovery for Cells (f)                   .0050
Minimum Recovery Justified for a Cell (f)         .350
Max Percent Solid for Concentrates (f)           .0100
Tolerance for Residence Time Calcul. (f)         .0100
Tolerance for Entrainment Calculation (f)        .0100
Tolerance for Circulating Flow Calcul. (f)           1
Max Iterations for Residence Time Calcul.            8
Max Iterations for Entrainment Calculation          10
Max Iterations for Circulating Flow Calcul.
```

```
                    ****** PRINT : EQUIPEMENT ******


     CIRCUIT                         Rghr
Number of Paral. Banks:                1
Number of Cells/Bank  :               13
Volume of Cells (m3)  :              7.99
                (pi3) :              282.0
Type of Cells:                      simple
Power Installed (HP):                20.0
Manufacturer(s) * :                   0
Flotation Time (min):                28.6
Unit Flotation Time :                2.20
Alternative Volume(m3):              4.25
                                     5.66
                                     8.50


   *  MANUFACTURER: (W)Wemco, (A)Agitair, (O) Outokumpu, (D)Dorr
```

```
                  ****** RESULTS : SUMMARY BALANCE ******

                            CIRCUIT : ROUGHER

                                      FLOWRATE t/h
                       New Feed       Rougher Conc          Tail
Solid                  100.000           5.900             94.100
Water                  138.095          20.795            117.300
Water Addition                           2.800
Water adjusment                           .000

                               GRADE  and DISTRIBUTION %
                        New Feed       Rougher Conc            Tail
                    Grade     Dist     Grade     Dist     Grade      Dist
ANTIMONY              4.7    100.0      58.1     73.6       1.3      26.4
QUARTZ               95.3    100.0      41.9      2.6      98.7      97.4
Water                58.0    100.0      77.9     15.1      55.5      84.9
Solid                42.0    100.0      22.1      5.9      44.5      94.1
```

APPENDIX 8

Preliminary Testing of Caribou Ore

PROCESS STUDIES GROUP

technical notes	*Project No.* E-76815	*File No.* E-14	
Project Beneficiation of Complex Sulphide Ores for Add-on Processing		*Date* 08-12-87	*Page* 1

Subject: Preliminary Testing of Caribou Ores.

Author(s): B. Blizzard

Objective: To determine if the Caribou ore is amenable to flotation using a bulk concentrate system, and to identify the problems with such a process.

Conclusions:

Author(s)	technical notes	Project No.	Page	of
B. Blizzard			2	3

Discussion: All tests were unsuccessful in producing a suitable concentrate grade for either lead or zinc (See Tests C-5 to C-15). Many of the tests produced no separation from the pyrite. More research was done to try to correct this and a number of problems became apparent.

1. The Caribou ore needs an extremely fine grind to liberate all of the minerals. A CANMET report states that a K80 of 12 µm is required.

2. Bulk flotation techniques are usually the same as selective flotation but are carried out at a lower pH (9.0 - 10.0). Pyrite will not float with xanthate as a collector above a pH of 10.0 but, it seems that when xanthate and thiocarbonate collectors are used together to enhance metal recoveries, the pyrite is recovered also. This seems to be a problem in these tests because there was too much material reporting to the concentrate. It is reported in a paper by S. M. Bulatouie and D. M. Wyslovzil of Lakefield Research that pyrite recovery to the concentrate can be as high as 50% (even while using SO_2 and NaCn as depressants at a pH of 10.0). This is because the froth becomes voluminous and unselective. In all tests, there were large amounts of froth in the roughers.

3. It has also been found that the types of pyrite most difficult to depress are those that come from either finely disseminated sulphides or pyrites with carbonaceous inclusions. The Caribou deposit falls into both of these categories.

4. The natural pH of the sample may also pose a problem. When the sample was first received, the natural pH of the ore was 13.0 when removed from the rod mill. Sulphuric acid had to be used to adjust the pH for the flotation tests. As the ore oxidized, the pH lowered on its own until the rod mill discharge was 4.0. then lime had to be used to adjust the pH. The high pH of the original samples would point to high carbonate content in the sample. The deposit normally does not have high carbonaceous content, which lead to the possibility that the sample may not be representative of the whole orebody.

5. A major problem with sulphide flotation is the effect of oxidation-reduction of "Eh". The precise meaning of Eh is obscure but is measured as the potential of a platinum electrode with respect to a calomel electrode. A grinding mill generates low Eh values when grinding sulphide ore and this results in depression of valuable minerals. That is why most plants processing complex sulphides aerate the pulp for long periods of time to raise the Eh. All tests were aerated for a period of 15 minutes using the Denver flotation cell, but the Eh was not measured at this time. For later empirical testing, the Eh will have to be the same for all tests.

Author(s) B. Blizzard	technical notes	Project No.	Page 3 of 3

Method: The ore sample was received from Caribou and was then crushed to -6 mesh. The crushed ore was split into one kilogram samples and bagged in preparation for grinding. For each flotation test, the rod mill was charged with two kilograms of ore and ground for 180 minutes producing a grind of K80 25 µm. A total of 11 flotation tests were performed.

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH

P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE:	Bulk Sulphide	PROJECT NUMBER:	E-76815
SAMPLE WEIGHT:		TEST NUMBER:	C-5
PARTICLE SIZE:	80% -25 μm	TEST BY:	B. Blizzard
PURPOSE:	Preliminary Test on Caribou Ore	DATE:	November 9, 1987
BACKGROUND:	Use CANMET Test A-30	CLIENT:	Caribou

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)						
					CuSO4	343	3894	31	Lime		
Grind	180										
Condition 1	10		9.8	Aerator	3.0				add to pH		
Condition 2	5		9.8	Aerator		0.05	0.04	0.05			
Rougher 1	3		9.8	2100		0.05	0.04				

COMMENTS: Floating too much material
1270 and 1271

Assays	%Cu	%Zn
Conc:	.54	5.44
Tails:	.25	4.03

TEST C-6

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Bulk Sulphide
SAMPLE WEIGHT:
PARTICLE SIZE: 80% -25 µm
PURPOSE: Float just the copper
BACKGROUND: Use only MIBC and xanthate

PROJECT NUMBER: H-76815
TEST NUMBER: C-6
TEST BY: B. Blizzard
DATE: November 10, 1987
CLIENT: Caribou

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)						
					343	MIBC					
Grind	180										
Condition 1	10			Aerator							
Condition 2	5			Aerator	0.05						
Rougher 1	2			2100	0.05	0.05					

COMMENTS: 1272 AND 1273

Assays	%Cu	%Zn
Conc:	.40	4.36
Tails:	.29	4.60

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE:	Bulk Sulphide	PROJECT NUMBER:	H-76815
SAMPLE WEIGHT:		TEST NUMBER:	C-7
PARTICLE SIZE:	80% -25 μm	TEST BY:	B. Blizzard
PURPOSE:	Preliminary Testing on Caribou Ore	DATE:	November 10, 1987
BACKGROUND:	Cut Cu in half and just use xanthate	CLIENT:	Caribou

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)						
					CuSo4	343	MIBC				
Grind	180										
Condition 1	10				1.5						
Condition 2	5					0.05					
Rougher 1	2			2100		0.05	0.05				

COMMENTS: 1274 AND 1275

Assays	%Cu	%Zn
Conc:	.31	4.71
Tails:	.31	4.71

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Bulk Sulphide
SAMPLE WEIGHT:
PARTICLE SIZE: 80% -25 μm
PURPOSE: Preliminary Testing on Caribou Ore
BACKGROUND: Repeat of Test C-5 using longer aeration time

PROJECT NUMBER: H-76815
TEST NUMBER: C-8
TEST BY: B. Blizzard
DATE: November 18, 1987
CLIENT: Caribou

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)						
					CuSo4	343	3894	31	H2SO4		
Grind	180										
Condition 1	30	Aerator			3.0				add to pH		
Condition 2	5	Aerator				0.05	0.04	0.05			
Rougher 1	3	2100	9.5			0.05	0.04				

COMMENTS: Floating too much material.

Assays	%Zn
Conc:	5.45
Tails:	0.99

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE:	Bulk Sulphide	**PROJECT NUMBER:**	H-76815
SAMPLE WEIGHT:		**TEST NUMBER:**	C-9
PARTICLE SIZE:	80% -25 µm	**TEST BY:**	B. Blizzard
PURPOSE:	Preliminary Testing on Caribou Ore	**DATE:**	November 18, 1987
BACKGROUND:	Using 3418A to suppress pyrite	**CLIENT:**	Caribou

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)						
					CuSo4	343	3418A	31	H2SO4		
Grind	180										
Condition 1	30				3.0						
Condition 2	5					0.05	0.04	0.05	add to pH		
Bulk Rougher	3		9.8	2100		0.05	0.04				

COMMENTS:	Assays	%Zn
	Conc:	5.34
	Tails:	0.63

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE:	Bulk Sulphide
SAMPLE WEIGHT:	
PARTICLE SIZE:	80% -25 µm
PURPOSE:	Preliminary Testing on Caribou Ore
BACKGROUND:	Use NaCn as pyrite depressant

PROJECT NUMBER:	H-76815
TEST NUMBER:	C-10
TEST BY:	B. Blizzard
DATE:	November 18, 1987
CLIENT:	Caribou

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)						
					CuSo4	343	3894	31	H2SO4	NaCn	
Grind	180										
Condition 1	30				3.0						
Condition 2	5					0.05	0.04	0.05	add to pH		
Bulk Rougher			9.6	2100		0.05	0.04			0.10	

COMMENTS:	Assays	%Zn
	Conc:	5.02
	Tails:	0.94

TEST C-11

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH

P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE:	Bulk Sulphide	**PROJECT NUMBER:**	H-76815
SAMPLE WEIGHT:		**TEST NUMBER:**	C-11
PARTICLE SIZE:	80% -25 µm	**TEST BY:**	B. Blizzard
PURPOSE:	Preliminary Testing on Caribou Ore	**DATE:**	November 26, 1987
BACKGROUND:	Use 3418A and add no xanthate to float	**CLIENT:**	Caribou

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)						
					CuSo4	350	3418A	31	20% H2SO4		
Grind	180								50 mL		
Condition 1	20				3.0						
Condition 2	5					0.05	0.04	0.05	add to pH 9.5		
Bulk Rougher	3		9.5	2100			0.04				

COMMENTS: Used 350 xanthate instead of 343

Assays	%Zn
Conc:	4.47
Tails:	4.75

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE:	Bulk Sulphide
SAMPLE WEIGHT:	
PARTICLE SIZE:	80% -25 µm
PURPOSE:	Preliminary Testing on Caribou Ore
BACKGROUND:	Use a cleaner to achieve better results
PROJECT NUMBER:	H-76815
TEST NUMBER:	C-12
TEST BY:	B. Blizzard
DATE:	November 27, 1987
CLIENT:	Caribou

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)						
					CuSo4	343	3418A	31	20% H2SO4		
Grind	180								40 mL		
Condition 1	30				3.0						
Condition 2	5					0.05	0.04	0.05	add to pH		
Rougher 1	3		9.7	2100		0.05	0.08				
Cleaner 1	1			1800							

COMMENTS: Cleaner floated in 500 g cell

Assays	%Zn
Conc:	3.45
Tails:	4.90

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH

P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE:	Bulk Sulphide	**PROJECT NUMBER:**	H-76815
SAMPLE WEIGHT:		**TEST NUMBER:**	C-13
PARTICLE SIZE:	80% -25 µm	**TEST BY:**	B. Blizzard
PURPOSE:	Preliminary Testing on Caribou Ore	**DATE:**	November 27, 1987
BACKGROUND:	Use SO2 as a pyrite depressant	**CLIENT:**	Caribou

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					
					CuSo4	3894	343	31	SO2	
Grind	180									
Condition 1	30								add to pH 6	condition air @ 1000 rpm
Condition 2	5					0.05		0.05		
Rougher 1	3		6.0			0.05				
Condition 3	5				1.0		0.05			add lime to pH 11.0
Rougher 2	3		11.0				0.05			

COMMENTS: Samples 1310-1312

Assays	%Zn	%Pb
Rougher 1:	5.89	2.21
Rougher 2:	3.47	
Tail:	5.14	

TEST C-14

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Bulk Sulphide
SAMPLE WEIGHT:
PARTICLE SIZE: 80% -25 μm
PURPOSE: Caribou Ore Testing
BACKGROUND: Use NaCn as pyrite depressant

PROJECT NUMBER: H-76815
TEST NUMBER: C-14
TEST BY: B. Blizzard
DATE: December 1, 1987
CLIENT: Caribou

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)						
					CuSo4	NaCn	343	Lime			
Grind	180										
Condition 1	30					4.0		add to pH			
Condition 2	5				1.4		0.05				
Rougher 1	3		9.5				0.05				

COMMENTS: Samples 1313-1316. Samples taken at 1 minute intervals

Assay	%Zn
Rougher 1:	3.94
2:	3.21
3:	3.41
Tail:	4.73

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH

P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE:	Bulk Sulphide	**PROJECT NUMBER:**	H-76815
SAMPLE WEIGHT:		**TEST NUMBER:**	C-15
PARTICLE SIZE:	80% -25 μm	**TEST BY:**	B. Blizzard
PURPOSE:	Preliminary Test Work	**DATE:**	December 7, 1987
BACKGROUND:	Use soda ash for pH and anaconda	**CLIENT:**	Caribou

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)						
					CuSo4	242	Soda Ash	343	Lime		
Grind	180										
Condition 1	30		9.5				adjust to pH				
CuPb Rougher	2	1800	9.5			0.015					
Condition 2	5				0.575			0.05	adjust to pH		
Zn Rougher	2		11.5								

COMMENTS: Samples 1317, 1318, 1319

Assays	%Zn	%Pb
Pb Conc:		
Zn Conc:		
Tail:		

LAKEFIELD RESEARCH

Semi-Quantitative Spectrographic Analysis

Distribution Mr. Brent Blizzard

Report No. Q 2325

Analytical Method: OE

Date December 30, 1987

Requested by:

Charge

Received From: New Brunswick Natural Resources

No. of Samples

Sample No. LR 8728769

Sample Description Pulp 87-1333

CARIBOU Sample low grade

10 - 100 %	Si
3 - 30 %	Fe(Zn)
1 - 10%	(Pb)
0.3 - 3 %	Mg
0.1 - 1 %	Cu
0.03 - 0.3 %	As
0.01 - 0.1 %	Sb
0.003 - 0.03 %	Cd
0.001 - 0.01 %	Sn, Mn, Bi, Al, Mo, Ag, Zr, Ca
0.0003 - 0.003 %	Cr
0.0001 - 0.001 %	-
<0.0003 %	
I	
S	Pb, Zn

I = Interference prevents positive identification
S = Strong spectral lines, unable to estimate amount

Unless specified above, the following were not detected at the approximate ppm lower limits of 0.5 Cu, Ag; 1 Mn; 5 Mg, Cr; 10 Be, Bi, Ca, Co, Ni, V; 25 Ge, Fe, Pb, Mo, Si, Sr, Sn, Ti, Zr, Tl, Pd, U, Th; 50 Al, Sb, Bi, Cd, Ga, Li, Zn; 100 As, Au, Ba, In, Na; 200 Nb, Ta, W, Rb, Pt; 300 P, Te, Y, Ca; 1000 K.

A. E. Carr, Manager - Assay Servic

APPENDIX 9

Caribou Factorial Design Flotation Test
C-17 to C-20

PROCESS STUDIES GROUP	technical notes	Project No.	File No.
	Project Flotation	Date 9/11/88	Page 1

Subject: Caribou Factorial Design Flotation Test C-17 to C-20

Author(s): D. Taylor

Objective: To determine the effect of aeration, pH and xanthate levels on the lead flotation circuit, and pH, copper sulphate levels and xanthate levels on the zinc flotation circuit of Caribou ore. The testing was done using a fractional factorial design system to determine the importance of the parameters.

Conclusions: (1) The test showed that pH in the lead circuit was very important; however, aeration and xanthate additions within the range tested were not important for proper metallurgical performance. The tests showed that higher pH values are a definite asset to lead recovery. However, we find that circuits with higher pH values recover more zinc mineral in the lead concentrate, which is detrimental to overall metallurgical performance.

(2) The tests showed that pH, copper sulphate, and xanthate additions within the range were not important to the metallurgical performance of the zinc circuit.

Discussion: This series of tests was carried out on the basis of information supplied by Caribou Mines Ltd. in their mine feasibility data report (file No. 444-26-0002) and on information received from Brunswick Mining and Smelting on their procedure for laboratory flotation tests. From a study of the data available, it was determined that aeration, pH, and xanthate addition levels seemed to be significant parameters in the lead circuit. The levels were set at what appeared to be maximum and minimum levels (on the basis of the information received). Aeration times of 20 and 60 minutes, pH levels of 8.0 and 10.0, and xanthate levels of 50 and 100 g/tonne were used in the lead circuit. In the zinc circuit, pH levels of 10.2 and 10.8, copper sulphate levels of 700 and 1000 g/tonne, and xanthate levels of 70 to 100 g/tonne were tested as the result of information obtained.

A fractional factorial design test was constructed and the test work was completed. The data on the recovery of metal in the second concentrate fraction of each test was used in the mathematical analysis of the data. Using the "Probability Points" of the variance ratio (F - Distribution), with data from one test with four samples, and probability of 0.05 ($F_{1,4,0.05}$) = 7.71 and the variance ratio from the mathematical analysis, the significant parameters can be determined. Any variance ratio over 7.71 would be significant as a control parameter in the tests involved. In the data on the lead circuit, the variance ratio of pH was 12.34, while the other parameters were below 0.5, resulting in the pH being the only significant parameter. In the zinc circuit, the variance test was based on the results of zinc recovery in the zinc circuit on the second zinc concentrate. The highest variance ratio was 6.34 for copper sulphate with the other parameters below 1.0. Since all of the values were below the F test value of 7.71, no parameter was significant in the zinc circuit.

By examining the variance ratio and test results, it is apparent that circuits with higher pH values produce better lead recoveries within the range tested.

TEST NUMBER	PH	LEAD RECOVERY 2ND PB CONC	ZINC RECOVERY 2ND ZN CONC
C-17	8.0	49.98	26.30
C-18	8.0	59.77	28.35
C-19	10.0	83.17	61.84
C-20	10.0	82.09	52.02

Author(s)	technical notes	Project No.	Page	of
D. Taylor			3	4

Discussion: A significant problem resulted from the higher pH values in the lead circuit, that is, high quantities of zinc were recovered in the lead concentrate. The loss of zinc to the lead circuit would cause the zinc circuit to perform very poorly. It would make reasonable zinc grades and recoveries very difficult to obtain. This would result in poorer overall metallurgy for the entire circuit. For proper metallurgical performance, a recovery of approximately 15% of the zinc to the lead circuit would be reasonable. In test C-19, almost 62% of the zinc was recovered in the lead concentrate. Further laboratory testing is necessary to determine methods of lowering zinc recoveries in the lead concentrate.

Method: A fractional factorial design for the lead circuit was set up using the parameters shown below:

TEST NUMBER	AERATION TIME IN MINUTES	PH	XANTHATE G/TONNE
C-17	20	8.0	100
C-18	60	8.0	50
C-19	20	10.0	50
C-20	60	10.0	100

For the zinc circuit:

TEST NUMBER	PH	COPPER SULPHATE G/TONNE	XANTHATE G/TONNE
C-17	10.2	700	100
C-18	10.8	700	70
C-19	10.2	1000	70
C-20	10.8	1000	100

The tests were run by grinding approximately 2000 g of ore in the laboratory rod mill with 1000 mL of water, 150 g/tonne of SO_2, and 2 kg/tonne of sodium carbonate. The laboratory mill was run for 3 hours to produce the desired grind. The sample was removed from the mill and split by a bucket - mixer - beaker method into four approximately equal samples.

The samples were then placed in a 500 g stainless steel flotation cell and then placed on the Denver laboratory flotation machine. The tests were then run according to the flotation test reports (shown in Figures 1 to 4). The resultant flotation samples were filtered, dried, and assayed. The data was then placed on a standard metallurgical balance sheet (shown in Figures 5 to 8). The results from the metallurgical balances were then used in the fractional factorial design system to produce the coefficient of variance data, and to show the important parameters in the test.

FIGURE 1

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Caribou
SAMPLE WEIGHT: ~ 500 g
PARTICLE SIZE:
PURPOSE: Preliminary Factorial Design Test "Pb Float"
BACKGROUND:

PROJECT NUMBER: H-76815
TEST NUMBER: C-17
TEST BY: D. Taylor, et. al.
DATE: 5/7/88
CLIENT:

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)				aeration - 20 min. pH - 8.0 xanthate - 100 g/t
					80% Isopropyl, 20% Amyl Xanthate @ 1%	MIBC			
Aeration	20.0			1400 full					
pH Adjust			soda ash 8.0						
Xanthate					5 g @ 1%				
Condition	5.0			1400 off					
Float 1	0.5			2200		2 drops			
Float 2	0.5								
Float 3	1.0								
Float 4	1.0								
Float 5	2.0								
Float 6	2.0								

COMMENTS:

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Caribou
SAMPLE WEIGHT: ~ 500 g
PARTICLE SIZE:
PURPOSE: Preliminary Factorial Design Test "Zn Float"
BACKGROUND:

PROJECT NUMBER: H-76815
TEST NUMBER: C-17
TEST BY: D. Taylor, et. al.
DATE: 5/7/88
CLIENT:

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)				pH - 10.2 CuSO4 - 700 g/t xanthate - 100 g/t
					Lime	CuSO4	80% Isopropyl, 20% Amyl Xanthate @ 1%	MIBC	
pH Adjust			lime 10.2		x				
CuSo4						3.5 g @ 10%			
Condition	10.0			1400 off					
Xanthate & Con	5.0			1400 off			5 g @ 1%		
Float 1	0.5			2200				2 drops	
Float 2	0.5								
Float 3	1.0								
Float 4	1.0								
Float 5	2.0								
Float 6	2.0								

COMMENTS:

FIGURE 2

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Caribou
SAMPLE WEIGHT: 500 g
PARTICLE SIZE:
PURPOSE: Preliminary Factorial Design Test "Pb Float"
BACKGROUND:

PROJECT NUMBER: H-76815
TEST NUMBER: C-18
TEST BY: D. Taylor
DATE: 5/7/88
CLIENT:

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					aeration - 60 min pH - 8.0 xanthate - 50 g/t
					Mixture Xanthate	MIBC				
Aeration	60.0			1400 full						
Xanthate & pH			soda ash 8.0		2.5 g					
Condition	5.0			1400 off						
Float 1	0.5			2100		2 drops				
Float 2	0.5									
Float 3	1.0									
Float 4	1.0									
Float 5	2.0									
Float 6	2.0									

COMMENTS:

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Caribou
SAMPLE WEIGHT: 500 g
PARTICLE SIZE:
PURPOSE: Preliminary Factorial Design Test "En Float"
BACKGROUND:

PROJECT NUMBER: H-76815
TEST NUMBER: C-18
TEST BY: D. Taylor
DATE: 5/7/88
CLIENT:

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					pH - 10.8 CuSO4 - 700 g/t Xanthate - 70 g/t
					Lime	CuSO4	Xanthte	MIBC		
pH & CuSO4			Lime 10.8			3.5 g @ 10%				
Condition	10.0			1400 off						
Xanthate							3.5 g			
Condition	5.0			1400 off						
Float 1	0.5			2100				2 drops		
Float 2	0.5									
Float 3	1.0									
Float 4	1.0									
Float 5	2.0									
Float 6	2.0									

COMMENTS:

FIGURE 3

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE:	Caribou	**PROJECT NUMBER:**	H-76815
SAMPLE WEIGHT:	500 g	**TEST NUMBER:**	C-19
PARTICLE SIZE:		**TEST BY:**	D. Taylor
PURPOSE:	Preliminary Factorial Design "Pb Float"	**DATE:**	5/7/88
BACKGROUND:		**CLIENT:**	

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					aeration - 20 min. pH - 10.0 xanthate - 50 g/t
					Mixture Xanthate	MIBC				
Aeration	20.0			1400 full						
Xanthate & pH			soda ash 10		2.5 g					
Condition	5.0			1400 off						
Float 1	0.5			2100		2 drops				
Float 2	0.5									
Float 3	1.0									
Float 4	1.0									
Float 5	2.0									
Float 6	2.0									

COMMENTS:

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH

P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Caribou
SAMPLE WEIGHT: 500 g
PARTICLE SIZE:
PURPOSE: Preliminary Factorial Design Test "Zn Float"
BACKGROUND:

PROJECT NUMBER: H-76815
TEST NUMBER: C-19
TEST BY: D. Taylor
DATE: 5/7/88
CLIENT:

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					pH - 10.2 CuSO4 - 1000 g/t Xanthate - 70 g/t
					Lime	CuSO4	Mixture Xanthate	MIBC		
pH & CuSO4			Lime 10.2			5 g @ 10%				
Condition	10.0			1400 off						
Xanthate							3.5 g			
Condition	5.0			1400 off						
Float 1	0.5			2100				2 drops		
Float 2	0.5									
Float 3	1.0									
Float 4	1.0									
Float 5	2.0									
Float 6	2.0									

COMMENTS:

FIGURE 4

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE:	Caribou	**PROJECT NUMBER:**	H-76815
SAMPLE WEIGHT:	500 g	**TEST NUMBER:**	C-20
PARTICLE SIZE:		**TEST BY:**	D. Taylor
PURPOSE:	Preliminary Factorial Design "Pb Float"	**DATE:**	5/7/88
BACKGROUND:		**CLIENT:**	

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					aeration - 60 min. pH - 10.0 xanthate - 100 g/t
					Mixture Xanthate	MIBC				
Aeration	60.0			1400 full						
Xanthate & pH			soda ash 10		5 g					
Condition	5.0			1400 off						
Float 1	0.5			2100		2 drops				
Float 2	0.5									
Float 3	1.0									
Float 4	1.0									
Float 5	2.0									
Float 6	2.0									

COMMENTS:

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE:	Caribou	**PROJECT NUMBER:**	H-76815
SAMPLE WEIGHT:	500 g	**TEST NUMBER:**	C-20
PARTICLE SIZE:		**TEST BY:**	D. Taylor
PURPOSE:	Preliminary Factorial Design Test "En Float"	**DATE:**	5/7/88
BACKGROUND:		**CLIENT:**	

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					pH - 10.8 CuSO4 - 1000 g/t Xanthate - 100 g/t
					Lime	CuSO4	Mixture Xanthate	MIBC		
pH & CuSO4			Lime 10.8			5 g @ 10%				
Condition	10.0			1400 off						
Xanthate							5 g			
Condition	5.0			1400 off						
Float 1	0.5			2100				2 drops		
Float 2	0.5									
Float 3	1.0									
Float 4	1.0									
Float 5	2.0									
Float 6	2.0									

COMMENTS:

FIGURE 5

CARIBOU FLOTATION TEST C-17

METAL DISTRIBUITIONS

DATE : JULY 5,1988

SAMPLE NAME	SAMPLE WEIGHT (grams)	Wt. Percent	METAL Pb (%)	METAL Zn (%)	Pb DIST (%)	Zn DIST (%)	CUMUL Pb DIST	CUMUL Zn DIST
Pb CONC 1	75.5	16.60	10.10	8.24	36.04	18.87	36.04	18.87
Pb CONC 2	23.8	5.23	12.40	10.30	13.95	7.43	49.98	26.30
Pb CONC 3	13.4	2.95	11.10	13.50	7.03	5.49	57.01	31.79
Zn CONC 1	45.4	9.98	3.47	27.20	7.44	37.45	64.45	69.23
Zn CONC 2	35.9	7.89	3.61	14.30	6.12	15.57	70.58	84.80
Zn CONC 3	28.1	6.18	3.68	2.26	4.89	1.93	75.47	86.73
TAILS	232.8	51.18	2.23	1.88	24.53	13.3	100.00	100.00
	0.0	0.00	0.00	0.00	0.00	ERR	100.00	ERR
	454.9							

METAL CONTENT (Pb)	METAL CONTENT (Zn)	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL GRADE (Pb&Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
167.63	136.76	10.10	8.24	18.34	36.04	18.87
64.88	53.89	10.65	8.73	19.38	49.98	26.30
32.70	39.77	10.70	9.30	20.01	57.01	31.79
34.63	271.45	8.63	14.44	23.07	64.45	69.23
28.49	112.85	7.70	14.41	22.11	70.58	84.80
22.73	13.96	7.19	12.88	20.07	75.47	86.73
114.12	96.21	4.65	7.25	11.90	100.00	100.00
0.00	0.00	4.65	7.25	11.90	100.00	ERR
465.18	724.90					

ZINC CIRCUIT

	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
Zn CONC 1	3.47	27.20	17.32	54.90
Zn CONC 2	3.53	21.50	31.56	77.72
Zn CONC 3	3.57	16.56	42.93	80.54
TAILS	2.66	6.57	100.00	100.00

FIGURE 6

CARIBOU FLOTATION TEST C-18

METAL DISTRIBUITIONS

DATE : JULY 5,1988

SAMPLE NAME	SAMPLE WEIGHT (grams)	Wt. Percent	METAL Pb (%)	METAL Zn (%)	Pb DIST (%)	Zn DIST (%)	CUMUL Pb DIST	CUMUL Zn DIST
b CONC 1	123.1	23.56	10.10	8.55	49.98	23.42	49.98	23.42
b CONC 2	23.2	4.44	10.50	9.58	9.79	4.94	59.77	28.35
b CONC 3	19.2	3.67	10.10	9.17	7.80	3.91	67.57	32.27
Zn CONC 1	73.8	14.12	2.73	27.60	8.10	45.26	75.67	77.53
Zn CONC 2	25.4	4.86	4.64	18.70	4.74	10.55	80.40	88.08
Zn CONC 3	11.1	2.12	5.90	11.40	2.63	2.81	83.04	90.90
TAILS	246.8	47.23	1.71	1.66	16.95	9.1	100.00	100.00
	0.0	0.00	0.00	0.00	0.00	ERR	100.00	ERR
	522.6							

METAL CONTENT (Pb)	METAL CONTENT (Zn)	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL GRADE (Pb&Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
257.91	201.63	10.10	8.56	18.66	49.98	23.42
46.61	42.53	10.16	8.72	18.89	59.77	28.35
37.11	33.69	10.16	8.77	18.93	67.57	32.27
38.55	389.76	7.87	14.58	22.45	75.67	77.53
22.55	90.89	7.56	14.98	22.53	80.40	88.08
12.55	24.21	7.49	14.83	22.32	83.04	90.90
80.76	78.39	4.76	8.61	13.37	100.00	100.00
0.00	0.00	4.76	8.61	13.37	100.00	ERR
476.02	861.11					

ZINC CIRCUIT

	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
Zn CONC 1	2.73	27.60	24.97	66.82
Zn CONC 2	3.22	25.32	39.58	82.41
Zn CONC 3	3.49	23.92	47.69	86.55
TAILS	2.25	8.54	100.00	100.00

FIGURE 7

CARIBOU FLOTATION TEST C-19

METAL DISTRIBUITIONS

DATE : JULY 5,1988

SAMPLE NAME	SAMPLE WEIGHT (grams)	Wt. Percent	METAL Pb (%)	METAL Zn (%)	Pb DIST (%)	Zn DIST (%)	CUMUL Pb DIST	CUMUL Zn DIST
Pb CONC 1	201.2	36.34	8.44	9.26	55.78	38.80	55.78	38.80
Pb CONC 2	85.8	15.50	9.72	12.90	27.39	23.05	83.17	61.84
Pb CONC 3	35.5	6.41	5.31	12.80	6.19	9.45	89.36	71.31
Zn CONC 1	35.4	6.39	1.52	27.10	1.77	19.98	91.13	91.28
Zn CONC 2	30.9	5.58	2.19	9.44	2.22	6.07	93.35	97.36
Zn CONC 3	20.9	3.78	2.46	2.84	1.69	1.24	95.04	98.59
TAILS	143.9	25.99	1.05	0.47	4.96	1.4	100.00	100.00
	0.0	0.00	0.00	0.00	0.00	ERR	100.00	ERR
	553.6							

METAL CONTENT (Pb)	METAL CONTENT (Zn)	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL GRADE (Pb&Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
306.74	336.54	8.44	9.26	17.70	55.78	38.80
150.65	199.93	8.82	10.35	19.17	83.17	61.84
34.05	82.08	8.44	10.62	19.05	89.36	71.31
9.72	173.29	7.75	12.25	20.00	91.13	91.28
12.22	52.69	7.31	12.03	19.34	93.35	97.36
9.29	10.72	7.06	11.56	18.62	95.04	98.59
27.29	12.22	5.50	8.57	14.17	100.00	100.00
0.00	0.00	5.50	8.67	14.17	100.00	ERR
549.95	867.48					

ZINC CIRCUIT

	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
Zn CONC 1	1.52	27.10	16.61	69.62
Zn CONC 2	1.83	18.87	37.49	90.78
Zn CONC 3	1.98	15.03	53.36	95.09
TAILS	1.40	5.96	100.00	100.00

FIGURE 8

CARIBOU FLOTATION TEST C-20

METAL DISTRIBUITIONS DATE : JULY 5,1988

SAMPLE NAME	SAMPLE WEIGHT (grams)	Wt. Percent	METAL Pb (%)	METAL Zn (%)	Pb DIST (%)	Zn DIST (%)	CUMUL Pb DIST	CUMUL Zn DIST
Pb CONC 1	200.0	37.35	8.49	10.20	69.13	43.04	69.13	43.04
Pb CONC 2	30.9	5.77	10.30	14.70	12.96	9.58	82.09	52.62
Pb CONC 3	19.1	3.57	3.71	14.20	2.89	5.72	84.98	58.34
Zn CONC 1	71.2	13.30	1.30	22.00	3.77	33.04	88.75	91.38
Zn CONC 2	34.2	6.39	1.96	8.11	2.73	5.85	91.47	97.24
Zn CONC 3.	20.3	3.79	2.06	2.84	1.70	1.22	93.18	98.45
TAILS	159.6	29.82	1.05	0.46	6.82	1.5	100.00	100.00
	0.0	0.00	0.00	0.00	0.00	ERR	100.00	ERR
	535.3							

METAL CONTENT (Pb)	METAL CONTENT (Zn)	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL GRADE (Pb&Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
317.21	381.09	8.49	10.20	18.69	69.13	43.04
59.46	84.86	8.73	10.80	19.53	82.09	52.62
13.24	50.67	8.35	11.06	19.41	84.98	58.34
17.29	292.62	6.79	13.49	20.27	88.75	91.38
12.52	51.81	6.32	12.97	19.29	91.47	97.24
7.81	10.77	6.09	12.42	18.51	93.18	98.45
31.31	13.71	4.59	8.86	13.44	100.00	100.00
0.00	0.00	4.59	8.86	13.44	100.00	ERR
458.83	885.54					

ZINC CIRCUIT

	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
Zn CONC 1	1.30	22.00	25.08	79.32
Zn CONC 2	1.51	17.49	43.25	93.36
Zn CONC 3	1.60	15.13	54.58	95.28
TAILS	1.29	6.92	100.00	100.00

APPENDIX 10

Caribou Factorial Design Flotation Test
C-21 to C-24

PROCESS STUDIES GROUP

technical notes

Project No.		File No.	
Project Flotation		Date 22/11/88	Page 1

Subject: Caribou Factorial Design Flotation Test C-21 to C-24

Author(s): Donald S. Taylor

Objective: With the completion of the first tests on Caribou ore, it was determined that procedures must be found to lower the amount of zinc recovered in the lead concentrate. From a literature search, it was determined that sodium cyanide and zinc sulphate should be tested as zinc depressants in the lead circuit. It was also determined from the last tests that pH was an important factor, and this factor was reused in the fractional factorial design.

A test would also be completed on the zinc circuit to determine if sodium cyanide or temperature would affect the zinc grade. In the previous test, copper sulphate appeared important, and these three factors were used in the fractional factorial design.

Conclusions: (1) The tests showed that the sodium cyanide and zinc sulphate levels used in this test had no affect on the lead metallurgy. The only factor that showed an affect was the pH of the flotation circuit. The higher pH values produced lower lead recoveries within the range of pH tested. The zinc recovered in the lead circuit was only affected by the pH, with higher pH values producing lower zinc recoveries. The test results showed, however, that the amount of zinc recovered in the lead concentrate was not lowered to the 15% range, but remained in the 40 to 55% range.

(2) The tests showed that sodium cyanide, copper sulphate and temperatures had no affect on the zinc metallurgy within the range of the parameters tested in the zinc circuit.

Discussion: The results of the first tests on Caribou ore gave flotation data that had high recoveries of zinc in the lead concentrate. In an effort to reduce the zinc in the lead concentrate, a study of the literature on zinc depressants was carried out.[1,2] From the information received, it was decided to test sodium cyanide and zinc sulphate as zinc mineral depressants. The reagent dosage levels were set at what appeared to be maximum and minimum levels on the basis of the literature study. The quantities of reagents were set so that the zinc sulphate concentrations were twice the level of the sodium cyanide concentrations. The factorial design levels for zinc sulphate were 0 and 80 g/tonne, for sodium cyanide 0 and 40 g/tonne, and the pH was tested at 9.3 and 9.8.

In the zinc circuit, the problem was low grade zinc concentrates in the first series of tests. In an effort to improve zinc grades, a literature study was carried out, [3,4] and it was determined that temperature and sodium cyanide would be used in the factorial design. The levels of temperature were 20°C and 30°C, the sodium cyanide reagent dosage was 0 and 40g/tonne, and copper sulphate was at 700 and 1000 g/tonne.

A fractional factorial design was constructed and the test was completed. The data on the recovery of minerals in the second concentrate fraction of each test was used in the mathematical analysis of the data. The F-distribution was used for the analysis with F = 7.71. The analysis of the lead circuit showed that pH was the only parameter that had a significant effect on lead recovery. The higher the pH, the lower the lead recovery.

The zinc recovered in the lead concentrate was affected only by the pH parameter in the fractional factorial design experiment. The higher the pH, the lower the zinc recovery.

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[1] Zunkel, A.D., Boorman, R.S., Morris, A.E., Wesly, R.J. Complex Sulphides, Processing of Ores, Concentrates and By Products. (Warrendale, Pennsylvania; A.I.M.E., Metallurgical Society, 1985, p. 109)

[2] Fuerstenau, M.C. Ed. Flotation - A.M. Gaudin Memorial Volumes - Volume 1. (New York, New York; American Institute of Mining, Metallurgical and Petroleum Engineers Inc., 1976, pp. 431-448)(DNRE Paper No. 0401062)

[3] Zunkel, op. cit., pp. 102-109, 129

[4] Fuerstenau, op. cit., (DNRE Paper No. 0401063), pp. 468-481

Author(s) D. Taylor	***technical notes***	*Project No.*	*Page of* 3

Discussion:

TEST NUMBER	pH	LEAD RECOVERY 2nd Pb CONC.	ZINC RECOVERY 2nd Pb CONC.
C-21	9.3	84.74	55.23
C-22	9.8	75.04	42.10
C-23	9.3	83.14	49.73
C-24	9.8	82.53	49.25

The results of the test in the lead flotation circuit creates a problem. If we operate at low pH values, we have high lead recoveries and high zinc recoveries, and vice versa. However, what we desire is high lead recoveries and low zinc recoveries in the flotation operation, and these tests do not achieve this goal. Therefore, we will have to conduct further tests in order to depress the zinc recovered in the lead circuit.

The analyses of zinc recovered in the zinc circuit showed that no parameter tested had any affect on the recovery of zinc in the zinc circuit. Therefore, further testing will be necessary in this circuit.

Author(s) D. Taylor	***technical notes***	*Project No.*	*Page of* 4

Method: A fractional factorial design for the lead circuit was set up using the parameters shown below:

TEST NUMBER	pH	SODIUM CYANIDE G/TONNE	ZINC SULPHATE G/TONNE
C-21	9.3	0.0	80.0
C-22	9.8	0.0	0.0
C-23	9.3	40.0	0.0
C-24	9.8	40.0	80.0

For the zinc circuit·

TEST NUMBER	COPPER SULPHATE G/TONNE	TEMPERATURE °C	SODIUM CYANIDE G/TONNE
C-21	700	20	40.0
C-22	1000	20	0.0
C-23	700	30	0.0
C-24	1000	30	40.0

The tests were run by grinding approximately 2000 g of ore in the laboratory rod mill with 1000 mL of water, 150 g/tonne SO_2 and 2 kg/tonne sodium carbonate. The laboratory mill was run for three hours to produce the desired grind. The sample was removed from the mill and split by a bucket-mixer-beaker method into four approximately equal samples.

The samples were then placed in a 500 g stainless steel flotation cell, then placed on the Denver laboratory flotation machine. The tests were then run according to the flotation test reports (shown in figures 1 to 4). The resultant flotation samples were filtered, dried and assayed. The data was then placed on a standard metallurgical balance sheet (shown in figures 5 to 8). The results from the metallurgical balances were then used in the fractional factorial design system to produce the coefficient of variance data, and to show the important parameters in the test.

FIGURE 1

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Caribou
SAMPLE WEIGHT: 500 g
PARTICLE SIZE:
PURPOSE: Preliminary Factorial Design Test "Pb Float"
BACKGROUND:

PROJECT NUMBER: H-76815
TEST NUMBER: C-21
TEST BY: D. Taylor
DATE: 19/01/88
CLIENT:

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					pH @ 9.3 NaCN - 0.0 g/t ZnSO4 - 80.0 g/t
					NaCn @ 1%	ZnSO4 @ 1%	Xanthate Mixture	MIBC		
Aeration	20.0			1400 full						
pH			soda ash 9.3							
NaCN & ZnSO4					0.0	4 g				
Condition	5.0			1400 none						
Xanthate							2.5 g			
Condition	5.0			1400 none						
Float 1	1.0			2100				2 drops		
Float 2	2.0									
Float 3	4.0									

COMMENTS:

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Caribou
SAMPLE WEIGHT: 500 g
PARTICLE SIZE:
PURPOSE: Preliminary Factorial Design Test "En Float"
BACKGROUND:

PROJECT NUMBER: H-76815
TEST NUMBER: C-21
TEST BY: D. Taylor
DATE: 19/01/88
CLIENT:

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)						CuSO4 - 700 g/t temperature - 20°C NaCN - 40 g/t
					Lime	CuSO4	NaCN	Xanthate Mixture	Temp	MIBC	
pH & CuSO4			Lime 10.3			3.5 g					
Condition	10.0			1400 none							
NaCN & Temp.							2.0 g		20°C		
Condition	2.0			1400 none							
Xanthate								3.5 g			
Condition	5.0			1400 none							
Float 1	1.0			2100						2 drops	
Float 2	2.0										
Float 3	4.0										

COMMENTS:

FIGURE 2

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Caribou
SAMPLE WEIGHT: 500 g
PARTICLE SIZE:
PURPOSE: Preliminary Factorial Design Test "Pb Float"
BACKGROUND:

PROJECT NUMBER: H-76815
TEST NUMBER: C-22
TEST BY: D. Taylor
DATE: 19/01/88
CLIENT:

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					pH 0 9.8 NaCN - 0.0 g/t ZnSO4 - 0.0 g/t
					NaCn @ 1%	ZnSO4 @ 1%	Xanthate Mixture	MIBC		
Aeration	20.0			1400 full						
pH			soda ash 9.8							
NaCN & ZnSO4					0.0	0.0				
Condition	5.0			1400 none						
Xanthate							2.5 g			
Condition	5.0			1400 none						
Float 1	1.0			2100				2 drops		
Float 2	2.0									
Float 3	4.0									

COMMENTS:

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Caribou
SAMPLE WEIGHT: 500 g
PARTICLE SIZE:
PURPOSE: Preliminary Factorial Design Test "En Float"
BACKGROUND:

PROJECT NUMBER: H-76015
TEST NUMBER: C-22
TEST BY: D. Taylor
DATE: 19/01/88
CLIENT:

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					CuSO4 - 1000 g/t temperature - 20°C NaCN - 0.0 g/t
					CuSO4	NaCN	Temp	Xanthate Mixture	MIBC	
pH & CuSO4			Lime 10.3		5.0 g					
Condition	10.0			1400 none						
NaCN & Temp.						0.0	20°C			
Condition	2.0			1400 none						
Xanthate								3.5 g		
Condition	5.0			1400 none						
Float 1	1.0			2100					2 drops	
Float 2	2.0									
Float 3	4.0									

COMMENTS:

FIGURE 3

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Caribou
SAMPLE WEIGHT: 500 g
PARTICLE SIZE:
PURPOSE: Preliminary Factorial Design Test "Pb Float"
BACKGROUND:

PROJECT NUMBER: H-76815
TEST NUMBER: C-23
TEST BY: D. Taylor
DATE: 19/01/88
CLIENT:

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					pH - 9.3 NaCN - 40 g/t ZnSO4 - 0.0 g/t
					NaCn @ 1%	ZnSO4 @ 1%	Xanthate Mixture	MIBC		
Aeration	20.0			1400 full						
pH			soda ash 9.3							
NaCN & ZnSO4					2.0 g	0.0				
Condition	5.0			1400 none						
Xanthate							2.5 g			
Condition	5.0			1400 none						
Float 1	1.0			2100				2 drops		
Float 2	2.0									
Float 3	4.0									

COMMENTS:

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE:	Caribou	PROJECT NUMBER:	H-76815
SAMPLE WEIGHT:	500 g	TEST NUMBER:	C-23
PARTICLE SIZE:		TEST BY:	D. Taylor
PURPOSE:	Preliminary Factorial Design Test "Zn Float"	DATE:	19/01/88
BACKGROUND:		CLIENT:	

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					CuSO4 - 700 g/t temperature - 30°C NaCN - 0.0 g/t
					CuSO4	NaCN	Temp	Xanthate Mixture	MIBC	
pH & CuSO4			Lime 10.3		3.5 g					
Condition	10.0			1400 none						
NaCN & Temp.						0.0	30°C			
Condition	2.0			1400 none						
Xanthate								3.5 g		
Condition	5.0			1400 none						
Float 1	1.0			2100					2 drops	
Float 2	2.0									
Float 3	4.0									

COMMENTS:

FIGURE 4

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Caribou
SAMPLE WEIGHT: 500 g
PARTICLE SIZE:
PURPOSE: Preliminary Factorial Design Test "Pb Float"
BACKGROUND:

[illegible]ECT NUMBER: H-76815
TEST NUMBER: C-24
TEST BY: D. Taylor
DATE: 19/07/88
CLIENT:

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					pH - 9.8 NaCN - 40 g/t ZnSO4 - 40.0 g/t
					NaCn @ 1%	ZnSO4 @ 1%	Xanthate Mixture	MIBC		
Aeration	20.0			1400 full						
pH			soda ash 9.8							
NaCN & ZnSO4					2.0 g	4.0				
Condition	5.0			1400 none						
Xanthate							2.5 g			
Condition	5.0			1400 none						
Float 1	1.0			2100				2 drops		
Float 2	2.0									
Float 3	4.0									

COMMENTS:

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH

P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE:	Caribou	PROJECT NUMBER:	H-76815
SAMPLE WEIGHT:	500 g	TEST NUMBER:	C-24
PARTICLE SIZE:		TEST BY:	D. Taylor
PURPOSE:	Preliminary Factorial Design Test "Zn Float"	DATE:	19/07/88
BACKGROUND:		CLIENT:	

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					CuSO4 - 1000 g/t temperature - 30°C NaCN - 40.0 g/t
					CuSO4	NaCN	Temp	Xanthate Mixture	MIBC	
pH & CuSO4			Lime 10.3		5.0 g					
Condition	10.0			1400 none						
NaCN & Temp.						2.0	30°C			
Condition	2.0			1400 none						
Xanthate								3.5 g		
Condition	5.0			1400 none						
Float 1	1.0			2100					2 drops	
Float 2	2.0									
Float 3	4.0									

COMMENTS:

FIGURE 5

CARIBOU FLOTATION TEST C-21

METAL DISTRIBUITIONS   DATE : JULY 5,1988

SAMPLE NAME	SAMPLE WEIGHT (grams)	Wt. Percent	METAL Pb (%)	METAL Zn (%)	Pb DIST (%)	Zn DIST (%)	CUMUL Pb DIST	CUMUL Zn DIST
Pb CONC 1	173.0	44.36	8.38	8.93	72.77	47.50	72.77	47.50
Pb CONC 2	21.3	5.46	11.20	11.80	11.97	7.73	84.74	55.23
Pb CONC 3	9.9	2.54	6.38	12.50	3.17	3.81	87.91	59.03
Zn CONC 1	28.3	7.26	1.34	33.00	1.90	28.72	89.82	87.75
Zn CONC 2	20.4	5.23	2.02	12.90	2.07	8.09	91.89	95.84
Zn CONC 3	12.8	3.28	2.53	4.74	1.63	1.87	93.51	97.71
TAILS	124.3	31.87	1.04	0.60	6.49	2.3	100.00	100.00
	0.0	0.00	0.00	0.00	0.00	ERR	100.00	ERR
	390.0							

METAL CONTENT (Pb)	METAL CONTENT (Zn)	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL GRADE (Pb&Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
371.73	396.13	8.38	8.93	17.31	72.77	47.50
61.17	64.45	8.69	9.24	17.93	84.74	55.23
16.20	31.73	8.58	9.40	17.98	87.91	59.03
9.72	239.46	7.70	12.27	19.97	89.82	87.75
10.57	67.48	7.24	12.33	19.56	91.89	95.84
8.30	15.56	7.01	11.96	18.97	93.51	97.71
33.15	19.12	5.11	8.34	13.45	100.00	100.00
0.00	0.00	5.11	8.34	13.45	100.00	ERR
510.83	833.92					

ZINC CIRCUIT

	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
Zn CONC 1	1.34	33.00	15.75	70.10
Zn CONC 2	1.62	24.58	32.86	89.85
Zn CONC 3	1.81	20.45	46.31	94.40
TAILS	1.30	7.17	100.00	100.00

FIGURE 6

CARIBOU FLOTATION TEST C-22

METAL DISTRIBUITIONS DATE : JULY 5,1988

SAMPLE NAME	SAMPLE WEIGHT (grams)	Wt. Percent	METAL Pb (%)	METAL Zn (%)	Pb DIST (%)	Zn DIST (%)	CUMUL Pb DIST	CUMUL Zn DIST
Pb CONC 1	125.7	34.83	9.27	8.93	63.39	35.40	63.39	35.40
Pb CONC 2	18.8	5.21	11.40	11.30	11.66	6.70	75.04	42.10
Pb CONC 3	12.8	3.55	8.15	12.20	5.57	4.92	80.72	47.02
Zn CONC 1	54.1	14.99	2.11	24.10	6.21	41.12	86.93	88.14
Zn CONC 2	15.9	4.41	3.63	14.20	3.14	7.12	90.07	95.26
Zn CONC 3	14.2	3.93	3.02	4.87	2.33	2.18	92.40	97.44
TAILS	119.4	33.08	1.17	0.68	7.60	2.6	100.00	100.00
	0.0	0.00	0.00	0.00	0.00	ERR	100.00	ERR
	360.9							

METAL CONTENT (Pb)	METAL CONTENT (Zn)	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL GRADE (Pb&Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
322.87	311.03	9.27	8.93	18.20	63.39	35.40
59.38	58.86	9.55	9.24	18.79	75.04	42.10
28.91	43.27	9.43	9.48	18.91	80.72	47.02
31.63	361.27	7.56	13.22	20.78	86.93	88.14
15.99	62.56	7.28	13.29	20.57	90.07	95.26
11.88	19.16	7.03	12.79	19.83	92.40	97.44
38.71	22.50	5.09	8.79	13.88	100.00	100.00
0.00	0.00	5.09	8.79	13.88	100.00	ERR
509.37	878.65					

ZINC CIRCUIT

	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
Zn CONC 1	2.11	24.10	32.21	77.61
Zn CONC 2	2.46	21.85	48.49	91.05
Zn CONC 3	2.55	18.99	60.59	95.17
TAILS	1.74	8.25	100.00	100.00

FIGURE 7

CARIBOU FLOTATION TEST C-23

METAL DISTRIBUITIONS DATE : JULY 5,1988

SAMPLE NAME	SAMPLE WEIGHT (grams)	Wt. Percent	METAL Pb (%)	METAL Zn (%)	Pb DIST (%)	Zn DIST (%)	CUMUL Pb DIST	CUMUL Zn DIST
Pb CONC 1	166.4	43.61	8.58	8.68	75.46	45.23	75.46	45.23
Pb CONC 2	14.6	3.83	9.95	9.84	7.68	4.50	83.14	49.73
Pb CONC 3	8.5	2.23	7.56	12.20	3.40	3.25	86.53	52.97
Zn CONC 1	30.2	7.91	1.25	36.20	2.00	34.23	88.53	87.21
Zn CONC 2	14.0	3.67	2.46	18.00	1.82	7.89	90.35	95.10
Zn CONC 3	11.3	2.96	2.86	6.24	1.71	2.21	92.06	97.31
TAILS	136.6	35.80	1.10	0.63	7.94	2.7	100.00	100.00
	0.0	0.00	0.00	0.00	0.00	ERR	100.00	ERR
	381.6							

METAL CONTENT (Pb)	METAL CONTENT (Zn)	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL GRADE (Pb&Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
374.14	378.50	8.58	8.68	17.26	75.46	45.23
38.07	37.65	8.69	8.77	17.46	83.14	49.73
16.84	27.18	8.64	8.93	17.57	86.53	52.97
9.89	286.49	7.62	12.68	20.30	88.53	87.21
9.03	66.04	7.31	13.00	20.31	90.35	95.10
8.47	18.48	7.11	12.68	19.79	92.06	97.31
39.38	22.56	4.96	8.37	13.33	100.00	100.00
0.00	0.00	4.96	8.37	13.33	100.00	ERR
495.81	836.88					

ZINC CIRCUIT

	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
Zn CONC 1	1.25	36.20	14.82	72.79
Zn CONC 2	1.63	30.44	28.34	89.57
Zn CONC 3	1.88	25.51	41.02	94.27
TAILS	1.33	7.82	100.00	100.00

FIGURE 8

CARIBOU FLOTATION TEST C-24

METAL DISTRIBUITIONS

DATE : JULY 5,1988

SAMPLE NAME	SAMPLE WEIGHT (grams)	Wt. Percent	METAL Pb (%)	METAL Zn (%)	Pb DIST (%)	Zn DIST (%)	CUMUL Pb DIST	CUMUL Zn DIST
Pb CONC 1	145.9	42.25	8.27	8.71	72.22	43.37	72.22	43.37
Pb CONC 2	16.4	4.75	10.50	10.50	10.31	5.88	82.53	49.25
Pb CONC 3	9.8	2.84	6.86	11.90	4.02	3.98	86.56	53.23
Zn CONC 1	32.6	9.44	1.88	30.70	3.67	34.16	90.22	87.38
Zn CONC 2	12.1	3.50	2.93	16.60	2.12	6.85	92.35	94.24
Zn CONC 3	9.6	2.78	3.04	6.94	1.75	2.27	94.09	96.51
TAILS	118.9	34.43	0.83	0.86	5.91	3.5	100.00	100.00
	0.0	0.00	0.00	0.00	0.00	ERR	100.00	ERR
	345.3							

METAL CONTENT (Pb)	METAL CONTENT (Zn)	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL GRADE (Pb&Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
349.43	368.02	8.27	8.71	16.98	72.22	43.37
49.87	49.87	8.50	8.89	17.39	82.53	49.25
19.47	33.77	8.40	9.06	17.46	86.56	53.23
17.75	289.84	7.36	12.51	19.87	90.22	87.38
10.27	58.17	7.12	12.74	19.86	92.35	94.24
8.45	19.29	6.94	12.49	19.43	94.09	96.51
28.58	29.61	4.84	8.49	13.32	100.00	100.00
0.00	0.00	4.84	8.49	13.32	100.00	ERR
483.82	848.59					

ZINC CIRCUIT

	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
Zn CONC 1	1.88	30.70	27.29	73.02
Zn CONC 2	2.16	26.88	43.07	87.68
Zn CONC 3	2.32	23.36	56.06	92.54
TAILS	1.30	7.91	100.00	100.00

APPENDIX 11

Caribou Factorial Design Flotation Test
C-25 to C-28

PROCESS STUDIES GROUP

technical notes

Project No.

File No.

Project: Flotation

Date: 5/12/88

Page: 1

Subject: Caribou Factorial Design Flotation Test C-25 to C-28

Author(s): Donald S. Taylor

Objective: With the completion of the first two test runs on Caribou ore, it was determined that procedures must be found to lower the amount of zinc recovered in the lead concentrate. From a literature search, it was determined that aeration at low pH's with SO_2 gas, high quantities of sodium cyanide, and pH would be tested as zinc depressants.

A test would also be conducted on the zinc circuit to determine if potassium permanganate, high flotation temperatures, and copper sulphate addition levels would be parameters useful in producing better zinc metallurgy.

Conclusions:

1. The test results showed that the flotation pH, the aeration at low pH values with SO_2 gas, and sodium cyanide all had an affect on lead concentrate recoveries. The higher pH values produced generally better lead recoveries, the aeration at low pH values with SO_2 gas produced lower lead recoveries, and the use of sodium cyanide produced higher lead recoveries. The zinc recovered in the lead concentrate was affected by the pH and aeration at low pH values with SO_2 gas. The lower pH values resulted in poorer recovery of zinc, and the aeration with SO_2 gas at low pH produced poorer zinc recoveries. As in the previous test, the results of the lead and zinc recoveries in the lead circuit work against each other, as we desire high lead recoveries and low zinc recoveries, but any control parameter appears to do just the opposite. The tests showed, however, that the amount of zinc recovered in the lead concentrate was not lowered to the 15% range but was in the 26 to 47% range, which is an improvement over the last series of tests. The results of this data indicate more test work must be completed to lower the zinc recovery in the lead circuit.

2. The tests showed that temperature, potassium permanganate and copper sulphate additions all had an affect on the zinc metallurgy. The strongest affect was with temperature, where higher temperatures produced much poorer zinc recoveries. The potassium permanganate additions produced better recoveries and the use of smaller quantities of copper sulphate produced generally better recoveries.

Discussion: The results of the two tests completed on Caribou ore showed flotation results that had high recoveries of zinc in the lead concentrate. In an effort to reduce the zinc in the lead concentrate, a further study on zinc depressants was carried out.[1] From the information studied, it was decided to test aeration at low pH values with SO_2 gas, high quantities of sodium cyanide and pH as zinc depressants in the lead flotation. The fractional factorial design levels for sodium cyanide were 0 and 300 g/tonne, aeration was done at the natural pH from the grind or at a pH of 6.0 produced by using SO_2 gas sparged into the pulp during the aeration procedure. The pH range was narrowed down to units of 9.0 and 9.2, from data received during the last series of tests.

In the zinc circuit, the problem with low grade zinc concentrate was observed in the preceding series of tests. In a continuing effort to improve the zinc grade, a further literature study was carried out.[2,3] From the results of this study, it was determined that temperature, potassium permanganate and copper sulphate would be used in the factorial design. The levels of temperature were set at 20°C and 70°C, the potassium permanganate at 0 and 300 g/tonne, and copper sulphate at 700 and 1000 g/tonne.

A fractional factorial design was constructed and the test work was completed. The data on the recovery of the mineral in the second concentrate fraction of each test was used in the mathematical analysis of the data. The F-distribution test was used for the analysis of the test results with F = 7.71. The analysis of the circuit showed all parameters as being important. The most important parameter in the lead circuit was the aeration step, where the use of SO_2 gas to produce a pH of 6.0 during aeration caused poorer lead metallurgy. The second most important parameter would be sodium cyanide, which showed improved lead metallurgy when being used. The parameter of pH was also found significant in which the higher pH values, on average, produced better lead metallurgy.

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[1] Zunkel, A.D., Boorman, R.S., Morris, A.E., Wesly, R.J. Complex Sulphides, Processing of Ores, Concentrates and By Products. (Warrendale, Pennsylvania; A.I.M.E., Metallurgical Society, 1985, p. 103-129)

[2] Zunkel, op. cit., pp. 106, 124-129

[3] Fuerstenau, M.C. Ed. Flotation - A.M. Gaudin Memorial Volumes - Volume 1. (New York, New York; American Institute of Mining, Metallurgical and Petroleum Engineers Inc, 1976, pp. 481)(DNRE Paper No. 0401063)

Discussion:

TEST NUMBER	AERATION	LEAD RECOVERY 2ND PB CONC	ZINC RECOVERY 2ND ZN CONC
C-25	natural	79.59	40.37
C-26	natural	75.96	46.80
C-27	6.0 pH with SO_2	44.57	26.03
C-28	6.0 pH with SO_2	71.81	30.62

TEST NUMBER	SODIUM CYANIDE G/TONNE	LEAD RECOVERY 2ND PB CONC	ZINC RECOVERY 2ND ZN CONC
C-25	300.0	79.59	40.37
C-26	0.0	75.96	46.80
C-27	0.0	44.57	26.03
C-28	300.0	71.81	30.62

TEST NUMBER	PH	LEAD RECOVERY 2ND PB CONC	ZINC RECOVERY 2ND ZN CONC
C-25	9.0	79.59	40.37
C-26	9.2	75.96	46.80
C-27	9.0	44.57	26.03
C-28	9.2	71.81	30.62

Discussion: The zinc recovered in the lead circuit was affected by the aeration and pH parameters. The aeration at pH 5.0 with SO₂ gas showed that less zinc was recovered in the lead concentrate. The zinc recovery at 26 and 30% was not down to the projected [illegible]% levels, but they were below the levels of the previous tests in the 40 to 55% zinc recovery range. The use of lower pH values in the circuit resulted in lower amounts of zinc being recovered on the average in the lead circuit.

The test results of the lead circuit continued to cause problems in improving the overall metallurgy. If we operate with aeration at a pH of 6.0 by SO₂ gas, we get poor lead recoveries, which is detrimental, and we get poorer zinc recoveries, which is beneficial. The pH parameter is likewise contradictory, high pH values produce, on the average, better lead recoveries, which is beneficial, but higher pH values also recover more zinc in the lead circuit, which is detrimental. Therefore, the tests results show a no-win situation in lead metallurgy, and as a result, more test work is needed.

The analysis of the zinc recovered in the zinc circuit showed that all of the parameters had an effect. The parameter of temperature had the greatest affect, with higher temperatures producing very poor zinc metallurgy. The use of potassium permanganate as an oxidant showed that, on the average, the use of the oxidant produced better zinc recoveries. The parameters of copper sulphate showed that, on the average, the lower reagent dosages produced better zinc recoveries. It appears that further test work with these parameters needs to be carried out.

TEST NUMBER	TEMPERATURE °C	ZINC RECOVERY IN 2ND ZINC CONC.
C-25	70	17.91
C-26	20	64.00
C-27	20	83.00
C-28	70	19.35

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Discussion:

TEST NUMBER	POTASSIUM PERMANGANATE G/TONNE	ZINC RECOVERY IN 2ND ZINC CONC.
C-25	0.0	17.91
C-26	0.0	64.00
C-27	300.0	83.00
C-28	300.0	19.35

TEST NUMBER	COPPER SULPHATE G/TONNE	ZINC RECOVERY IN 2ND ZINC CONC.
C-25	700	17.91
C-26	1000	64.00
C-27	700	83.00
C-28	1000	19.35

Method: A fractional factorial design for the lead circuit was set up using these parameters, as shown below:

TEST NUMBER	PH	AERATION	SODIUM CYANIDE G/TONNE
C-25	9.0	natural	300.0
C-26	9.2	natural	0.0
C-27	9.0	6.0 pH with SO_2	0.0
C-28	9.2	6.0 pH with SO_2	300.0

For the zinc circuit:

TEST NUMBER	COPPER SULPHATE G/TONNE	POTASSIUM PERMANGANATE G/TONNE	TEMPERATURE °C
C-25	700	0.0	70
C-26	1000	0.0	20
C-27	700	300.0	70
C-28	1000	300.0	20

The tests were run by grinding approximately 2000 g of ore in the laboratory rod mill with 1000 mL of water, 150 g/tonne SO_2, and 2 kg/tonne sodium carbonate. The laboratory mill was run for three hours to produce the desired grind. The sample was removed from the mill and split by a bucket-mixer-beaker method into four approximately equal samples.

The samples were then placed in a 500 g stainless steel flotation cell, then placed on the Denver laboratory flotation machine. The tests were then run according to the flotation test reports (shown in figures 1 to 4). The resultant flotation samples were filtered, dried and assayed. The data was then placed on a standard metallurgical balance sheet (shown in figures 5 to 8). The results from the metallurgical balances were then used in the fractional factorial design system to produce the coefficient of variance data, and to show the important parameters in the test.

FIGURE 1

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE:	Caribou	**PROJECT NUMBER:**	H-76815
SAMPLE WEIGHT:	500 g	**TEST NUMBER:**	C-25
PARTICLE SIZE:		**TEST BY:**	D. Taylor
PURPOSE:	Preliminary Factorial Design Test "Pb Float"	**DATE:**	16/08/88
BACKGROUND:		**CLIENT:**	

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					pH - 9.0 aeration - natural NaCN - 300 g/t
					NaCN @ 1%	Xanthate Mixture	MIBC			
Aeration	20.0		natural	1400 full						
pH			soda ash 9.0							
NaCN					15 g					
Condition	5.0			1400 none						
Xanthate						2.5 g				
Condition	5.0			1400 none						
Float 1	1.0			2100 full			2 drops			
Float 2	2.0									
Float 3	4.0									

COMMENTS: natural pH 8.16

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Caribou
SAMPLE WEIGHT: 500 g
PARTICLE SIZE:
PURPOSE: Preliminary Factorial Design Test "Sn Float"
BACKGROUND:

PROJECT NUMBER: H-76815
TEST NUMBER: C-25
TEST BY: D. Taylor
DATE: 16/08/88
CLIENT:

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					$CuSO_4$ - 700 g/t $KMnO_4$ & air - none Temperature - 70°C
					$KMnO_4$ @ 1%	$CuSO_4$	Temperature	Xanthate Mixture	MIBC	
pH			lime 10.3	1400 none						
$KMnO_4$ & air	10.0			1400 none						
pH & $CuSO_4$			lime 10.3			3.5 g				
Condition	5.0									
Temperature							70°C			
Xanthate								3.5 g		
Condition	5.0			2100						
Float 1	1.0								2 drops	
Float 2	2.0									
Float 3	4.0									

COMMENTS: Sn float was very poor, bubbles but no solids

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE:	Caribou	PROJECT NUMBER:	H-76815
SAMPLE WEIGHT:	500 g	TEST NUMBER:	C-26
PARTICLE SIZE:		TEST BY:	D. Taylor
PURPOSE:	Preliminary Factorial Design Test "Pb Float"	DATE:	16/08/88
BACKGROUND:		CLIENT:	

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					pH - 9.2 aeration - natural NaCN - 0.0
					NaCN @ 1%	Xanthate Mixture	MIBC			
Aeration	20.0		natural	1400 full						
pH			soda ash 9.2							
NaCN										
Condition	5.0			1400 none						
Xanthate						2.5 g				
Condition	5.0			1400 none						
Float 1	1.0			2100			2 drops			
Float 2	2.0									
Float 3	4.0									

COMMENTS:

FIGURE 2

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Caribou
SAMPLE WEIGHT: 500 g
PARTICLE SIZE:
PURPOSE: Preliminary Factorial Design Test "Zn Float"
BACKGROUND:

PROJECT NUMBER: H-76815
TEST NUMBER: C-26
TEST BY: D. Taylor
DATE: 16/08/88
CLIENT:

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					CuSO4 - 1000 g/t KMNO4 & air - none Temperature - 20°C
					KMNO4 @ 1%	CuSO4	Temperature	Xanthate Mixture	MIBC	
pH			lime 10.3	1400 none						
KMNO4 & air	10.0			1400 none						
pH & CuSO4						5.0 g				
Condition	5.0			1400 none						
Temperature							20°C			
Xanthate								3.5 g		
Condition	5.0									
Float 1	1.0			2100					2 drops	
Float 2	2.0									
Float 3	4.0									

COMMENTS:

FIGURE 3

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE:	Caribou	**PROJECT NUMBER:**	H-76815
SAMPLE WEIGHT:	500 g	**TEST NUMBER:**	C-27
PARTICLE SIZE:		**TEST BY:**	D. Taylor
PURPOSE:	Preliminary Factorial Design Test "Pb Float"	**DATE:**	16/08/88
BACKGROUND:		**CLIENT:**	

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					pH - 9.0 aeration @ 6.0 with SO2 NaCN - none
					NaCN @ 1%	Xanthate Mixture	MIBC			
Aeration	20.0		SO2 6.0	1400 full						
pH			soda ash 9.0							
NaCN										
Condition	5.0			1400 none						
Xanthate						2.5 g				
Condition	5.0			1400 none						
Float 1	1.0			2100			2 drops			
Float 2	2.0									
Float 3	4.0									

COMMENTS:

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE:	Caribou	**PROJECT NUMBER:**	H-76815
SAMPLE WEIGHT:	500 g	**TEST NUMBER:**	C-27
PARTICLE SIZE:		**TEST BY:**	D. Taylor
PURPOSE:	Preliminary Factorial Design Test "Zn Float"	**DATE:**	16/08/88
BACKGROUND:		**CLIENT:**	

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					CuSO4 - 700 g/t KMNO4 & air - 300 g/t Temperature - 20°C
					KMNO4 @ 1%	CuSO4	Temperature	Xanthate Mixture		
pH			lime 10.3	1400 none						
KMNO4 & air	10.0			1400 full	15 g					
pH & CuSO4			lime 10.3			3.5 g				
Condition	5.0			1400 none						
Temperature							20°C			
Xanthate								3.5 g		
Condition	5.0									
Float 1	1.0			2100						
Float 2	2.0									
Float 3	4.0									

COMMENTS: Zn float was very poor, bubbles but no solids

FIGURE 4

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Caribou
SAMPLE WEIGHT: 500 g
PARTICLE SIZE:
PURPOSE: Preliminary Factorial Design Test "Pb Float"
BACKGROUND:

PROJECT NUMBER: H-76815
TEST NUMBER: C-28
TEST BY: D. Taylor
DATE: 16/06/88
CLIENT:

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					pH - 9.2 aeration - 6.0 with SO2 NaCN - 300 g/t
					NaCN @ 1%	Xanthate Mixture	MIBC			
Aeration	20.0		SO2 6.0	1400 full						
pH			soda ash 9.2							
NaCN *					15 g					
Condition	5.0			1400 none						
Xanthate						2.5 g				
Condition	5.0			1400 none						
Float 1	1.0			2100			2 drops			
Float 2	2.0									
Float 3	4.0									

COMMENTS: * MAKE SURE pH IS 9.2 BEFORE ADDING NaCN!!

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH

P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE:	Caribou	**PROJECT NUMBER:**	H-76815
SAMPLE WEIGHT:	500 g	**TEST NUMBER:**	C-28
PARTICLE SIZE:		**TEST BY:**	D. Taylor
PURPOSE:	Preliminary Factorial Design Test "Zn Float"	**DATE:**	16/08/88
BACKGROUND:		**CLIENT:**	

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					CuSO4 - 1000 g/t KMNO4 & air - 300 g/t Temperature - 70°C
					KMNO4 @ 1%	CuSO4	Temperature	Xanthate Mixture	MIBC	
pH			lime 10.3	1400 none						
KMNO4 & air	10.0			1400 full	15 g					
pH & CuSO4			lime 10.3			5.0 g				
Condition	5.0			1400 none						
Temperature							70°C			
Xanthate	5.0			1400 none				3.5 g		
Condition	1.0									
Float 1	2.0			2100					2 drops	
Float 2	4.0									
Float 3										

COMMENTS: Zn float was very poor, bubbles but no solids

FIGURE 5

CARIBOU FLOTATION TEST C-25

METAL DISTRIBUITIONS

DATE : JULY 5,1988

SAMPLE NAME	SAMPLE WEIGHT (grams)	Wt. Percent	METAL Pb (%)	METAL Zn (%)	Pb DIST (%)	Zn DIST (%)	CUMUL Pb DIST	CUMUL Zn DIST
Pb CONC 1	171.5	37.34	9.49	8.44	69.83	36.15	69.83	36.15
Pb CONC 2	18.8	4.09	12.10	8.97	9.76	4.21	79.59	40.37
Pb CONC 3	9.2	2.00	9.36	9.10	3.69	2.09	83.29	42.46
Zn CONC 1	9.7	2.11	1.92	30.70	0.80	7.44	84.09	49.89
Zn CONC 2	4.1	0.89	2.72	28.00	0.48	2.87	84.56	52.76
Zn CONC 3	2.0	0.44	2.97	9.91	0.25	0.50	84.82	53.26
TAILS	244.0	53.12	1.45	7.67	15.18	46.7	100.00	100.00
	0.0	0.00	0.00	0.00	0.00	ERR	100.00	ERR
	459.3							

METAL CONTENT (Pb)	METAL CONTENT (Zn)	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL GRADE (Pb&Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
354.35	315.14	9.49	8.44	17.93	69.83	36.15
49.53	36.72	9.75	8.49	18.24	79.59	40.37
18.75	18.23	9.73	8.52	18.25	83.29	42.46
4.05	64.84	9.37	9.55	18.92	84.09	49.89
2.43	24.99	9.24	9.90	19.14	84.56	52.76
1.29	4.32	9.18	9.90	19.09	84.82	53.26
77.03	407.46	5.07	8.72	13.79	100.00	100.00
0.00	0.00	5.07	8.72	13.79	100.00	ERR
507.43	871.70					

ZINC CIRCUIT

	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
Zn CONC 1	1.92	30.70	4.78	12.93
Zn CONC 2	2.16	29.90	7.64	17.91
Zn CONC 3	2.26	27.37	9.17	18.77
TAILS	1.50	8.87	100.00	100.00

FIGURE 5

CARIBOU FLOTATION TEST C-26

METAL DISTRIBUITIONS

DATE : AUGUST 22,1988

SAMPLE NAME	SAMPLE WEIGHT (grams)	Wt. Percent	METAL Pb (%)	METAL Zn (%)	Pb DIST (%)	Zn DIST (%)	CUMUL Pb DIST	CUMUL Zn DIST
Pb CONC 1	193.4	38.18	8.67	9.09	63.79	38.75	63.79	38.75
Pb CONC 2	32.0	6.32	10.00	11.40	12.17	8.04	75.96	46.80
Pb CONC 3	13.1	2.59	7.63	12.70	3.80	3.67	79.76	50.46
Zn CONC 1	47.3	9.34	2.39	22.90	4.30	23.88	84.07	74.34
Zn CONC 2	16.2	3.20	3.59	25.60	2.21	9.14	86.28	83.48
Zn CONC 3	10.7	2.11	3.81	18.20	1.55	4.29	87.83	87.78
TAILS	193.9	38.27	1.65	2.86	12.17	12.2	100.00	100.00
	0.0	0.00	0.00	0.00	0.00	ERR	100.00	ERR
	506.6							

METAL CONTENT (Pb)	METAL CONTENT (Zn)	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL GRADE (Pb&Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
330.99	347.02	8.67	9.09	17.76	63.79	38.75
63.17	72.01	8.86	9.42	18.28	75.96	46.80
19.73	32.84	8.79	9.60	18.39	79.76	50.46
22.31	213.81	7.73	11.80	19.53	84.07	74.34
11.48	81.86	7.51	12.54	20.05	86.28	83.48
8.05	38.44	7.38	12.73	20.12	87.83	87.78
63.15	109.47	5.19	8.95	14.14	100.00	100.00
0.00	0.00	5.19	8.95	14.14	100.00	ERR
518.88	895.45					

ZINC CIRCUIT

	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
Zn CONC 1	2.39	22.90	21.25	48.20
Zn CONC 2	2.70	23.59	32.19	66.66
Zn CONC 3	2.86	22.81	39.85	75.32
TAILS	1.98	8.38	100.00	100.00

FIGURE 7

CARIBOU FLOTATION TEST C-27

METAL DISTRIBUITIONS DATE : AUGUST 22,1988

SAMPLE NAME	SAMPLE WEIGHT (grams)	Wt. Percent	METAL Pb (%)	METAL Zn (%)	Pb DIST (%)	Zn DIST (%)	CUMUL Pb DIST	CUMUL Zn DIST
Pb CONC 1	96.4	20.42	7.67	7.38	31.03	18.54	31.03	18.54
Pb CONC 2	34.5	7.31	9.35	8.34	13.54	7.50	44.57	26.03
Pb CONC 3	19.2	4.07	10.70	12.30	8.62	6.15	53.19	32.19
Zn CONC 1	63.7	13.49	6.62	26.00	17.70	43.15	70.89	75.34
Zn CONC 2	9.5	2.01	5.67	17.40	2.26	4.31	73.15	79.65
Zn CONC 3	14.5	3.07	5.49	12.00	3.34	4.53	76.49	84.18
TAILS	234.4	49.64	2.39	2.59	23.51	15.8	100.00	100.00
	0.0	0.00	0.00	0.00	0.00	ERR	100.00	ERR
	472.2							

METAL CONTENT (Pb)	METAL CONTENT (Zn)	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL GRADE (Pb&Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
156.58	150.66	7.67	7.38	15.05	31.03	18.54
68.31	60.93	8.11	7.63	15.75	44.57	26.03
43.51	50.01	8.44	8.23	16.67	53.19	32.19
89.30	350.74	7.90	13.52	21.42	70.89	75.34
11.41	35.01	7.81	13.69	21.49	73.15	79.65
16.86	36.85	7.66	13.59	21.25	76.49	84.18
118.64	128.57	5.05	8.13	13.17	100.00	100.00
0.00	0.00	5.05	8.13	13.17	100.00	ERR
504.61	812.77					

ZINC CIRCUIT

	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
Zn CONC 1	6.62	26.00	37.81	63.64
Zn CONC 2	6.50	24.88	42.64	69.99
Zn CONC 3	6.33	22.75	49.77	76.67
TAILS	3.46	8.08	100.00	100.00

FIGURE 8

CARIBOU FLOTATION TEST C-28

METAL DISTRIBUITIONS

DATE : AUGUST 22,1988

SAMPLE NAME	SAMPLE WEIGHT (grams)	Wt. Percent	METAL Pb (%)	METAL Zn (%)	Pb DIST (%)	Zn DIST (%)	CUMUL Pb DIST	CUMUL Zn DIST
Pb CONC 1	134.5	30.61	10.60	7.45	62.87	26.45	62.87	26.45
Pb CONC 2	20.6	4.69	9.84	7.66	8.94	4.17	71.81	30.62
Pb CONC 3	15.5	3.53	7.33	7.43	5.01	3.04	76.82	33.66
Zn CONC 1	5.2	1.18	2.35	24.80	0.54	3.40	77.36	37.06
Zn CONC 2	9.5	2.16	1.39	37.60	0.58	9.43	77.94	46.49
Zn CONC 3	10.4	2.37	2.17	20.30	1.00	5.57	78.94	52.07
TAILS	243.7	55.46	1.96	7.45	21.06	47.9	100.00	100.00
	0.0	0.00	0.00	0.00	0.00	ERR	100.00	ERR
	439.4							

METAL CONTENT (Pb)	METAL CONTENT (Zn)	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL GRADE (Pb&Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
324.47	228.04	10.60	7.45	18.05	62.87	26.45
46.13	35.91	10.50	7.48	17.98	71.81	30.62
25.86	26.21	10.21	7.47	17.68	76.82	33.66
2.78	29.35	9.98	7.99	17.96	77.36	37.06
3.01	81.29	9.54	9.50	19.04	77.94	46.49
5.14	48.05	9.15	10.08	19.22	78.94	52.07
108.71	413.19	5.16	8.62	13.78	100.00	100.00
0.00	0.00	5.16	8.62	13.78	100.00	ERR
516.08	862.05					

ZINC CIRCUIT

	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
Zn CONC 1	2.35	24.80	2.32	5.13
Zn CONC 2	1.73	33.07	4.84	19.35
Zn CONC 3	1.91	27.78	9.13	27.75
TAILS	1.96	9.35	100.00	100.00

APPENDIX 12

Caribou Factorial Design Flotation Test C-29 to C-32

PROCESS STUDIES GROUP

technical notes	Project No.	File No.
Project Flotation	Date 19/12/88	Page 1

Subject: Caribou Factorial Design Flotation Test C-29 to C-32

Author(s): Donald S. Taylor

Objective: With the completion of the first three test runs on Caribou ore, it was determined that procedures must be found to lower the amount of zinc recovered in the lead concentrate. From the last series of tests, it was found that pH and aeration had very significant affects on both lead and zinc recovery in the lead concentrate. The continuing literature search added a new parameter to be tested; the use of a sodium sulphite-sodium sulphide reagent mix combination. These three parameters would be used in the testing done on the lead circuit in an attempt to lower zinc recoveries to reasonable ranges.

A continued search of the literature showed the possibility of improved metallurgical performance by changing the amount of SO_2 added to the grind. The SO_2 addition for this series of tests was therefore changed from 150 g/tonne to 950 g/tonne, based on the literature study.

A test would also be completed on the zinc circuit using the last test parameters of copper sulphide and potassium permanganate, and a new test parameter of very high pH values. This is done in a continuing attempt to produce better zinc metallurgy in the zinc circuit.

Conclusions: 1. The tests showed that aeration, pH and sodium sulphite-sodium sulphide parameters all had an affect on lead metallurgy. The use of natural aeration produced much better lead recoveries than using SO_2 gas during aeration at a pH of 6.0. The higher the pH of the lead flotation, the better the lead recovery. Lead recovery was also improved by the use of the sodium sulphite-sodium sulphide reagent combination. The zinc recovered in the lead concentrate was also affected by two of the tested parameters. The use of aeration with SO_2 gas lowered the amount of zinc recovered in the lead concentrate. The pH parameter showed that lower pH values were also able to produce lower recoveries of zinc in the lead circuit. The reagent combination of sodium sulphite-sodium sulphide had no affect on the zinc metallurgy. These tests showed that the amount of zinc recovered in the lead concentrate can be below 15% zinc. However, the lead recoveries for these low zinc values were only 13 and 35% lead in the lead circuit. This is a completely unacceptable situation, and further test work to obtain low zinc recoveries, with high lead recoveries must be pursued.

2. The use of higher quantities of SO_2 in the grind showed that the addition of more SO_2 inhibits the recovery of lead and the recovery of zinc in the lead concentrate. The net affect is that the higher quantities of SO_2 are detrimental to overall metallurgy.

3. The tests on the zinc circuit showed that pH and copper sulphate had an affect on zinc metallurgy. The lower the pH values and the lesser amounts of copper sulphate both produced better zinc recoveries. The high pH values increased the zinc grade substantially but with lower recoveries, this is a poor metallurgical situation. Test work needs to be continued in this circuit.

Discussion: The results of the first three tests completed on Caribou ore showed flotation results that had high recoveries of zinc in the lead concentrate. In an effort to reduce the zinc in the lead concentrate, a further literature study of zinc depressants was carried out.[1] The parameter of a combination of a sodium sulphite-sodium sulphide reagent was to be tested, along with two parameters from the last test (aeration and pH). The factorial design levels for sodium sulphite-sodium sulphide were 0.0 and 550 g/tonne, aeration was to be done at the natural pH from the grind, or a pH of 6.0 produced by SO_2 gas sparged into the pulp during the aeration procedure. The pH values set were at 9.0 and 9.2 based on the previous test.

In the zinc circuit, the problem was with poor zinc metallurgy (as shown in the previous sets of tests). In a continuing effort to produce better metallurgy, an examination of literature was continued[2], and the main parameter change would be the operation of the circuit at higher pH values. The parameters to be tested would be copper sulphate at 700 and 1000 g/tonne, pH conditions of 10.2 and 11.5, and potassium permanganate levels of 0.0 and 300 g/tonne.

This test included a change in grind reagents as the SO_2 in the grind was increased from 150 g/tonne to 950 g/tonne.[3] The data is compared from the last set of tests and this test under the same reagent conditions and shows that the use of higher quantities of SO_2 inhibits the recovery of lead and zinc. This is detrimental to overall metallurgy, and as a result, all further tests will be carried out at an SO_2 level of 150 g/tonne.

TEST NUMBER	SO2 TO GRIND G/TONNE	AERATION	LEAD RECOVERY IN 2ND PB CONC	ZINC RECOVERY IN 2ND ZN CONC
C-26	150	natural	75.96	46.80
C-30	950	natural	71.04	33.69
C-27	150	6.0 pH with SO2	44.57	26.03
C-29	950	6.0 pH with SO2	12.86	6.06

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[1] Zunkel, A. D., Boorman, R. S., Morris, A. E., Wesley, R. J. Complex Sulphides Processing of Ore, Concentrates and By Products, (Warrendale, Pennsylvania; A.I.M.E., Metallurgical Society), 1985, p. 100.

[2] Zunkel, op. cit., p. 124.

[3] Zunkel, op. cit., p. 105.

Author(s)	technical notes	Project No.	Page	of
D. Taylor			4	7

Discussion: A fractional factorial design was constructed and the test work was completed. The data on the recovery of mineral in the second concentrate fraction of each test was used in the mathematical analysis. The F-test distribution was used in the analysis of the test results with the F parameter being 7.71. The analysis of the circuit showed all parameters as being important in the recovery of lead in the lead circuit. The most important parameter for lead recovery was the aeration step, with natural aeration producing much better lead recoveries. The second most important parameter was pH, with higher pH values producing improved lead recovery. On average, the use of the combination reagent, sodium sulphite-sodium sulphide, produced slightly better lead recoveries. Overall lead metallurgy remained poor with low grade lead concentrates and poor recoveries of lead Normal lead recoveries in this circuit should be greater than 90%.

TEST NUMBER	AERATION	LEAD RECOVERY IN 2ND PB CONC	ZINC RECOVERY IN 2ND PB CONC
C-29	6.0 pH with SO2	12.86	6.06
C-30	natural	71.04	33.69
C-31	natural	65.07	27.15
C-32	6.0 pH with SO2	36.40	14.03

TEST NUMBER	PH	LEAD RECOVERY IN 2ND PB CONC	ZINC RECOVERY IN 2ND PB CONC
C-29	9.0	12.86	6.06
C-30	9.2	71.04	33.69
C-31	9.0	65.04	27.15
C-32	9.2	36.40	14.03

Discussion:

TEST NUMBER	SODIUM SULPHATE SODIUM SULPHIDE G/TONNE	LEAD RECOVERY IN 2ND PB CONC	ZINC RECOVERY IN 2ND PB CONC
C-29	0 + 0	12.86	6.06
C-30	0 + 0	71.04	33.69
C-31	550 + 550	65.09	27.15
C-32	550 + 550	36.40	14.03

The zinc recovered in the lead circuit was affected by the aeration and pH parameters. The aeration at a pH of 6.0 with SO_2 gas showed that significantly less zinc was recovered in the lead concentrate, than with natural aeration. The zinc recovered at the 6 to 14 percent zinc range is below our projected target of 15% zinc recovery in the lead concentrate. The use of lower pH's in the circuit resulted in lower amounts of zinc being recovered, on the average, in the lead circuit.

The test results of the lead circuit continue to cause problems in improving overall metallurgy. If we operate with aeration at a pH of 6.0 by SO_2 gas, we get poor lead recoveries, which is detrimental, and we get poorer zinc recoveries, which is beneficial. The pH parameter is likewise contradictory, higher pH values produce, on the average, better lead recoveries, which is good, but higher zinc recoveries, which is bad. Therefore, the test results show a no-win situation in lead metallurgy, and as a result, more test work is needed.

The analysis of the zinc recovered in the zinc circuit showed that two of the parameters had an affect on zinc metallurgy. The parameter of pH had the greatest affect, with higher pH values producing very poor zinc metallurgy. The parameter of copper sulphate showed that, on the average, the lower reagent dosage produced better zinc recoveries. It appears that further test work needs to be carried out on the parameters of pH and copper sulphate.

Author(s)	technical notes	Project No.	Page	of
D. Taylor			6	7

Discussion:

TEST NUMBER	PH	ZINC RECOVERED IN 2ND ZINC CONC.
C-29	10.2	74.55
C-30	10.2	71.45
C-31	11.5	48.27
C-32	11.5	38.97

TEST NUMBER	COPPER SULPHATE G/TONNE	ZINC RECOVERED IN 2ND ZINC CONC.
C-29	700	74.55
C-30	1000	71.45
C-31	700	48.27
C-32	1000	38.97

Author(s)		Project No.	Page	of
D. Taylor	technical notes		7	7

Method: A fractional factorial design for the lead circuit was set up using the parameters shown below:

TEST NUMBER	PH	SODIUM SULPHITE SODIUM SULPHIDE G/TONNE	AERATION
C-29	9.0	0 + 0	6.0 pH with SO2
C-30	9.2	0 + 0	natural
C-31	9.0	550 + 550	natural
C-32	9.2	550 + 550	6.0 pH with SO2

For the zinc circuit:

TEST NUMBER	COPPER SULPHATE G/TONNE	PH	POTASSIUM PERMANGANATE G/TONNE
C-29	700	10.2	300.0
C-30	1000	10.2	0.0
C-31	700	11.5	0.0
C-32	1000	11.5	300.0

The tests were run by grinding approximately 2000 g of ore in the laboratory rod mill with 1000 mL of water, 950 g/tonne SO_2, and 2 kg/tonne sodium carbonate. The laboratory mill was run for three hours to produce the desired grind. The sample was removed from the mill and split by a bucket-mixer-beaker method into four approximately equal samples.

The samples were then placed in a 500 g stainless steel flotation cell, then placed on the Denver laboratory flotation machine. The tests were then run according to the flotation test reports (shown in figures 1 to 4). The resultant flotation samples were filtered, dried and assayed. The data was then placed on a standard metallurgical balance sheet (shown in figures 5 to 8). The results from the metallurgical balances were then used in the fractional factorial design system to produce the coefficient of variance data, and to show the important parameters in the test.

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE:	Caribou	**PROJECT NUMBER:**	H-76815
SAMPLE WEIGHT:	500 g	**TEST NUMBER:**	C-29
PARTICLE SIZE:		**TEST BY:**	D. Taylor
PURPOSE:	Preliminary Factorial Design Test "Pb Float"	**DATE:**	26/08/88
BACKGROUND:		**CLIENT:**	

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					pH - 9.0 Na2S + Na2SO3 - 0 + 0 aeration - SO2 pH 6
					Na2S Na2SO3	Xanthate Mixture	MIBC			
Aeration	20.0		SO2 6.0	1400 full						
pH			soda ash 9.0							
Na2S & NaSO4										
Condition	5.0			1400 none						
Xanthate						2.5 g				
Condition	5.0			1400 none						
Float 1	1.0			2100 full			2 drops			
Float 2	2.0									
Float 3	4.0									

COMMENTS: First float poor, no froth tendency
Second float better, no lead though

FIGURE 1

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Caribou
SAMPLE WEIGHT: 500 g
PARTICLE SIZE:
PURPOSE: Preliminary Factorial Design Test "En Float"
BACKGROUND:

PROJECT NUMBER: H-76815
TEST NUMBER: C-29
TEST BY: D. Taylor
DATE: 26/08/88
CLIENT:

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					CuSO4 - 700 g/t pH - 10.2 KMNO4 & air - 300 g/t
					1% KMNO4	10% CuSO4	Xanthate Mixture	MIBC		
pH			Lime 10.2	1400 none						
KMNO4 + air	10.0			1400 full	15 g					
pH & CuSO4			Lime 10.2			3.5 g				
Condition	5.0			1400 none						
Xanthate	5.0			1400 none			3.5 g			
Float 1	1.0			2100				2 drops		
Float 2	2.0									
Float 3	4.0									

COMMENTS:

FIGURE 2

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE:	Caribou	PROJECT NUMBER:	H-76815
SAMPLE WEIGHT:	500 g	TEST NUMBER:	C-30
PARTICLE SIZE:		TEST BY:	D. Taylor
PURPOSE:	Preliminary Factorial Design Test "Pb Float"	DATE:	26/08/88
BACKGROUND:		CLIENT:	

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					pH - 9.2 Na2S + Na2SO3 - 0 + 0 aeration - natural
					Na2S Na2SO3	Xanthate Mixture	MIBC			
Aeration	20.0		natural	1400 full						
pH			soda ash 9.2							
Na2S & Na2SO4										
Condition	5.0			1400 none						
Xanthate						2.5 g				
Condition	5.0			1400 none						
Float 1	1.0			2100 full			2 drops			
Float 2	2.0									
Float 3	4.0									

COMMENTS:

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE:	Caribou	**PROJECT NUMBER:**	H-76815
SAMPLE WEIGHT:	500 g	**TEST NUMBER:**	C-29
PARTICLE SIZE:		**TEST BY:**	D. Taylor
PURPOSE:	Preliminary Factorial Design Test "En Float"	**DATE:**	26/08/88
BACKGROUND:		**CLIENT:**	

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					$CuSO_4$ - 1000 g/t pH - 10.2 $KMnO_4$ & air - none
					1% $KMnO_4$	10% $CuSO_4$	Xanthate Mixture	MIBC		
pH			Lime 10.2	1400 none						
$KMnO_4$ + air	10.0			1400 none						
pH & $CuSO_4$			Lime 10.2			5.0 g				
Condition	5.0									
Xanthate							3.5 g			
Condition	5.0			1400 none						
Float 1	1.0			2100 full				2 drops		
Float 2	2.0									
Float 3	4.0									

COMMENTS:
natural pH - 8.1
950 g SO_2 + 2 kg/tonne $NaCO_3$ in grind

FIGURE 3

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE:	Caribou	**PROJECT NUMBER:**	H-76815
SAMPLE WEIGHT:	500 g	**TEST NUMBER:**	C-31
PARTICLE SIZE:		**TEST BY:**	D. Taylor
PURPOSE:	Preliminary Factorial Design Test "Pb Float"	**DATE:**	26/08/88
BACKGROUND:		**CLIENT:**	

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					pH - 9.0 Na2S + Na2SO3 - 550 + 550 g/t aeration - natural
					Na2S 1% Na2SO3	Xanthate Mixture	MIBC			
Aeration	20.0			1400 full						
pH			soda ash 9.0							
Na2S & NaSO4					27.5 g/27.5 g					
Condition	5.0			1400 none						
Xanthate						2.5 g				
Condition	5.0									
Float 1	1.0			2100			2 drops			
Float 2	2.0									
Float 3	4.0									

COMMENTS:

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH

P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE:	Caribou	**PROJECT NUMBER:**	H-76815
SAMPLE WEIGHT:	500 g	**TEST NUMBER:**	C-31
PARTICLE SIZE:		**TEST BY:**	D. Taylor
PURPOSE:	Preliminary Factorial Design Test "Zn Float"	**DATE:**	26/08/88
BACKGROUND:		**CLIENT:**	

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					CuSO4 - 700 g/t pH - 11.5 KMNO4 & air - none
					1% KMNO4	10% CuSO4	Xanthate Mixture	MIBC		
pH			Lime 11.0	1400 none						
KMNO4 + air	10.0			1400 full						
pH & CuSO4			Lime 11.0			3.5 g				
Condition	5.0			1400 none						
Xanthate							3.5 g			
Condition	5.0			1400 none						
Float 1	1.0			2100				2 drops		
Float 2	2.0									
Float 3	4.0									

COMMENTS:

FIGURE 4

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Caribou
SAMPLE WEIGHT: 500 g
PARTICLE SIZE:
PURPOSE: Preliminary Factorial Design Test "Pb Float"
BACKGROUND:

PROJECT NUMBER: H-76815
TEST NUMBER: C-32
TEST BY: D. Taylor
DATE: 26/08/88
CLIENT:

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					pH - 9.2 Na2S + Na2SO3 - 550 + 550 g/t aeration - SO2 pH 6.0
					1% Na2S Na2SO3	Xanthate Mixture	MIBC			
Aeration	20.0		SO2 6.0	1400 full						
pH			soda ash 9.2							
Na2S & NaSO4					27.5 g/27.5 g					
Condition	5.0			1400 none						
Xanthate						2.5 g				
Condition	5.0			1400 none						
Float 1	1.0			2100 full			2 drops			
Float 2	2.0									
Float 3	4.0									

COMMENTS: First float poor
Second float much better
Third float barren

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE:	Caribou	**PROJECT NUMBER:**	H-76815
SAMPLE WEIGHT:	500 g	**TEST NUMBER:**	C-32
PARTICLE SIZE:		**TEST BY:**	D. Taylor
PURPOSE:	Preliminary Factorial Design Test "Zn Float"	**DATE:**	26/08/88
BACKGROUND:		**CLIENT:**	

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					CuSO4 - 1000 g/t pH - 11.5 KMNO4 & air - 300 g/t
					1% KMNO4	10% CuSO4	Xanthate Mixture	MIBC		
pH			Lime 11.0	1400 none						
KMNO4 + air	10.0			1400 full	15 g					
pH & CuSO4			Lime 11.0			5.0 g				
Condition	5.0									
Xanthate							3.5 g			
Condition	5.0			1400 none						
Float 1	1.0			2100				2 drops		
Float 2	2.0									
Float 3	4.0									

COMMENTS:

FIGURE 5

CARIBOU FLOTATION TEST C-29

METAL DISTRIBUITIONS

DATE : SEPTEMBER 01,1988

SAMPLE NAME	SAMPLE WEIGHT (grams)	Wt. Percent	METAL Pb (%)	METAL Zn 0.00	Pb DIST (%)	Zn DIST (%)	CUMUL Pb DIST	CUMUL Zn DIST
Pb CONC 1	10.2	1.80	6.62	7.65	2.65	1.55	2.65	1.55
Pb CONC 2	28.9	5.11	9.01	7.83	10.21	4.50	12.86	6.06
Pb CONC 3	32.5	5.75	10.40	9.13	13.25	5.91	26.11	11.96
Zn CONC 1	97.4	17.23	3.89	28.80	14.86	55.82	40.97	67.78
Zn CONC 2	26.5	4.69	6.63	18.60	6.89	9.81	47.86	77.59
Zn CONC 3	10.9	1.93	7.78	13.10	3.33	2.84	51.18	80.44
TAILS	358.8	63.48	3.47	2.74	48.82	19.6	100.00	100.00
	0.0	0.00	0.00	0.00	0.00	ERR	100.00	ERR
	555.2							

METAL CONTENT (Pb)	METAL CONTENT (Zn)	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL GRADE (Pb&Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
11.95	13.81	6.62	7.65	14.27	2.65	1.55
46.07	40.04	8.39	7.78	16.17	12.86	6.06
59.80	52.50	9.30	8.39	17.69	26.11	11.96
67.04	496.31	6.18	20.15	26.34	40.97	67.78
31.09	87.21	6.24	19.94	26.19	47.86	77.59
15.00	25.26	6.32	19.58	25.91	51.18	80.44
220.28	173.94	4.51	8.89	13.40	100.00	100.00
0.00	0.00	4.51	8.89	13.40	100.00	ERR
451.23	889.06					

ZINC CIRCUIT

	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
Zn CONC 1	3.89	28.80	20.11	63.41
Zn CONC 2	4.48	26.62	29.43	74.55
Zn CONC 3	4.74	25.53	33.93	77.78
TAILS	3.82	8.96	100.00	100.00

FIGURE .6

CARIBOU FLOTATION TEST C-30

METAL DISTRIBUITIONS

DATE : SEPTEMBER 01,1988

SAMPLE NAME	SAMPLE WEIGHT (grams)	Wt. Percent	METAL Pb (%)	METAL Zn (%)	Pb DIST (%)	Zn DIST (%)	CUMUL Pb DIST	CUMUL Zn DIST
Pb CONC 1	163.5	30.10	9.29	8.48	63.72	29.02	63.72	29.02
Pb CONC 2	22.1	4.07	7.89	10.10	7.31	4.67	71.04	33.69
Pb CONC 3	11.7	2.15	7.00	13.40	3.44	3.28	74.47	36.98
Zn CONC 1	63.4	11.67	1.00	26.60	2.66	35.30	77.13	72.28
Zn CONC 2	24.2	4.46	2.41	19.20	2.45	9.73	79.58	82.00
Zn CONC 3	16.1	2.96	3.02	14.00	2.04	4.72	81.62	86.72
TAILS	242.1	44.58	1.81	2.62	18.38	13.3	100.00	100.00
	0.0	0.00	0.00	0.00	0.00	ERR	100.00	ERR
	543.1							

METAL CONTENT (Pb)	METAL CONTENT (Zn)	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL GRADE (Pb&Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
279.68	255.29	9.29	8.48	17.77	63.72	29.02
32.11	41.10	9.12	8.67	17.80	71.04	33.69
15.08	28.87	9.00	8.95	17.95	74.47	36.98
11.67	310.52	7.05	13.24	20.30	77.13	72.28
10.74	85.55	6.66	13.75	20.41	79.58	82.00
8.95	41.50	6.46	13.76	20.23	81.62	86.72
80.69	116.79	4.39	8.80	13.19	100.00	100.00
0.00	0.00	4.39	8.80	13.19	100.00	ERR
438.91	879.63					

ZINC CIRCUIT

	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
Zn CONC 1	1.00	26.60	10.42	55.01
Zn CONC 2	1.39	24.56	20.00	71.45
Zn CONC 3	1.64	22.92	27.99	78.93
TAILS	1.76	8.71	100.00	100.00

FIGURE 7

CARIBOU FLOTATION TEST C-31

METAL DISTRIBUITIONS

DATE : SEPTEMBER 01,1988

SAMPLE NAME	SAMPLE WEIGHT (grams)	Wt. Percent	METAL Pb (%)	METAL Zn (%)	Pb DIST (%)	Zn DIST (%)	CUMUL Pb DIST	CUMUL Zn DIST
Pb CONC 1	145.9	23.16	9.33	7.95	55.51	23.08	55.51	23.08
Pb CONC 2	21.9	3.48	10.70	9.34	9.55	4.07	65.07	27.15
Pb CONC 3	13.6	2.16	9.35	11.20	5.19	3.03	70.26	30.18
Zn CONC 1	41.8	6.63	1.39	30.30	2.37	25.20	72.63	55.38
Zn CONC 2	11.4	1.81	1.66	37.50	0.77	8.51	73.40	63.88
Zn CONC 3	4.9	0.78	2.42	31.00	0.48	3.02	73.88	65.90
TAILS	390.5	61.98	1.64	4.26	26.12	33.1	100.00	100.00
	0.0	0.00	0.00	0.00	0.00	ERR	100.00	ERR
	630.0							

METAL CONTENT (Pb)	METAL CONTENT (Zn)	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL GRADE (Pb&Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
216.07	184.11	9.33	7.95	17.28	55.51	23.08
37.20	32.47	9.51	8.13	17.64	65.07	27.15
20.18	24.18	9.50	8.36	17.86	70.26	30.18
9.22	201.04	7.98	12.47	20.45	72.63	55.38
3.00	67.86	7.67	13.69	21.35	73.40	63.88
1.88	24.11	7.56	14.04	21.60	73.88	65.90
101.65	264.05	3.89	7.98	11.87	100.00	100.00
0.00	0.00	3.89	7.98	11.87	100.00	ERR
389.21	797.82					

ZINC CIRCUIT

	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
Zn CONC 1	1.39	30.30	7.97	36.09
Zn CONC 2	1.45	31.84	10.56	46.27
Zn CONC 3	1.53	31.77	12.19	52.60
TAILS	1.63	7.82	100.00	100.00

FIGURE 8

CARIBOU FLOTATION TEST C-32

METAL DISTRIBUITIONS

DATE : SEPTEMBER 01,1988

SAMPLE NAME	SAMPLE WEIGHT (grams)	Wt. Percent	METAL Pb (%)	METAL Zn (%)	Pb DIST (%)	Zn DIST (%)	CUMUL Pb DIST	CUMUL Zn DIST
Pb CONC 1	45.6	7.47	9.78	6.87	17.81	6.45	17.81	6.45
Pb CONC 2	43.9	7.19	10.60	8.39	18.59	7.58	36.40	14.03
Pb CONC 3	24.6	4.03	10.80	11.70	10.61	5.92	47.01	19.95
Zn CONC 1	37.2	6.09	1.93	31.00	2.87	23.73	49.88	43.68
Zn CONC 2	9.7	1.59	2.23	37.40	0.86	7.47	50.74	51.15
Zn CONC 3	4.3	0.70	2.95	25.70	0.51	2.27	51.25	53.42
TAILS	445.5	72.94	2.74	5.08	48.75	46.6	100.00	100.00
	0.0	0.00	0.00	0.00	0.00	ERR	100.00	ERR
	610.8							

METAL CONTENT (Pb)	METAL CONTENT (Zn)	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL GRADE (Pb&Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
73.01	51.29	9.78	6.87	16.65	17.81	6.45
76.19	60.30	10.18	7.62	17.80	36.40	14.03
43.50	47.12	10.32	8.50	18.81	47.01	19.95
11.75	188.80	8.25	14.03	22.28	49.88	43.68
3.54	59.39	7.89	15.44	23.33	50.74	51.15
2.08	18.09	7.76	15.70	23.47	51.25	53.42
199.85	370.52	4.10	7.95	12.05	100.00	100.00
0.00	0.00	4.10	7.95	12.05	100.00	ERR
409.92	795.52					

ZINC CIRCUIT

	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
Zn CONC 1	1.93	31.00	5.41	29.65
Zn CONC 2	1.99	32.32	7.04	38.97
Zn CONC 3	2.07	31.77	8.00	41.82
TAILS	2.67	7.83	100.00	100.00

APPENDIX 13

Caribou Factorial Design Flotation Test
C-33 to C-36

PROCESS STUDIES GROUP	technical notes	Project No.	File No.	
	Project Flotation		Date 16/01/89	Page 1

Subject: Caribou Factorial Design Flotation Test C-33 to C-36

Author(s): Donald S. Taylor

Objective: With the completion of the first four tests on Caribou ore, it was determined that procedures must be found to lower the amount of zinc recovered in the lead concentrate without decreasing the metallurgical performance of the lead circuit. From the last series of tests, it was found that pH had a very significant affect on both lead and zinc recovery in the lead concentrate. The continuing literature search added two new possible parameters; the use of a dithiophosphate collector (instead of a xanthate type collector) and the use of an organic depressant DS-20. These three parameters were used in the testing done on the lead circuit in an attempt to lower the zinc recoveries to a reasonable range.

A test was also completed on the zinc circuit using the last tests' parameters of copper sulphate and pH. A new parameter (the use of an organic depressant DS-20) was added to the factorial design in a continuing attempt to produce better zinc metallurgy.

Conclusions: The tests indicated that both pH and the use of the organic depressant DS-20 had an affect on lead metallurgy. The higher pH value produced better lead metallurgy, while, on the average, the use of the organic depressant DS-20 also improved lead metallurgy. The zinc recovered in the lead concentrate was only affected by the pH, with lower pH's producing less zinc recovery. These tests showed that the amount of zinc recovered in the lead concentrate (29-35%) was still far above the 15% recovery of zinc expected in the lead concentrate.

The tests showed that both the pH and the organic depressant DS-20 had an affect on zinc metallurgy. On average, the high pH values produced poor zinc recoveries, while the use of the organic depressant DS-20 produced better recoveries. However, the zinc recoveries are still low, therefore, test work needs to continue in the zinc circuit.

Discussion: The results of the first four tests completed on Caribou ore showed flotation results that had high recoveries of zinc in the lead concentrate. In an effort to reduce the zinc in the lead concentrate, a further study of depressants was carried out.[1] A literature search indicated that a different type of collector might help in zinc depression, and this, along with an organic depressant, was used for two of the parameters in the factorial design. The test on collectors was carried out on the regular collector used in all of the previous tests of 20% amyl xanthate and 80% isopropyl xanthate, and the use of R-242 (an amyl dithiophosphoric acid promoter). The organic depressant used was DS-20, consisting of two parts wheat dextrin, one part Quebracho, to two parts Lignosol (made at a 1% solution strength). The third test parameter for the factorial design was pH, set at ranges in line with the previous tests. The factorial design level for the collector was 50 g/tonne, for the organic depressant it was 0 and 30 g/tonne, and for pH the range tested was 8.5 and 9.5.

In the zinc circuit, the problem was the poor zinc metallurgy (as shown in the previous set of tests). In a continuing effort to produce better metallurgy, an examination of literature was continued,[2] and the use of an organic depressant DS-20 was tested. Therefore, the factorial design was set using pH, copper sulphate, and DS-20 as the tests parameters. The levels of the test were: the organic depressant at 0 and 30 g/tonne, the copper sulphate at 700 and 1000 g/tonne, and pH at 10 and 11.5.

A fractional factorial design was constructed and the test work was completed. The data on the recovery of mineral in the second concentrate fraction of each test was used in the mathematical analysis. The F test distribution was used for the analysis of the tests results with F for the test design being at 7.71. The analysis of the lead circuit showed only two parameters being important for lead recovery. The most important parameter was pH, which indicated better lead metallurgy with higher pH's. The second parameter found effective was the use of DS-20. This depressant caused lower lead recoveries in the lead circuit, which is not beneficial to overall metallurgical performance.

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[1] Zunkel, A.D., Boorman, R.S., Morris, A.E., Wesley, R.J. Complex Sulphides, Processing of Ores, Concentrates and By Products. (Warrendale, Pennsylvania; A.I.M.E., Metallurgical Society), 1985, p. 107-117, 124.

[2] Zunkel, op. cit., p. 112-117

Author(s) D. Taylor	**technical notes**	*Project No.*	*Page* 3 *of* 5

Discussion:

TEST NUMBER	PH	LEAD RECOVERED IN 2ND PB CONC.	ZINC RECOVERED IN 2ND PB CONC.
C-33	8.5	60.98	30.88
C-34	9.5	70.31	35.36
C-35	9.5	65.09	29.24
C-36	8.5	59.04	32.87

TEST NUMBER	DS-20 G/TONNE	LEAD RECOVERED IN 2ND PB CONC.	ZINC RECOVERED IN 2ND PB CONC.
C-33	0	60.98	30.88
C-34	0	70.31	35.36
C-35	30	65.09	29.24
C-36	30	59.04	32.87

The zinc recovered in the lead circuit was only affected by the pH parameter. The test data showed that higher pH values recovered more zinc in the lead circuit. The zinc recovered at 29-35% zinc was well above the projected value of 15% zinc in the lead concentrate, which was expected.

The test results of the lead circuit continue to cause problems in improving overall metallurgy. If we operate at high pH values, we get good lead recovery, which is beneficial, but also good zinc recovery, which is detrimental. The contradictory nature of the lead and zinc in the lead concentrate continues to occur, therefore, more test work needs to be completed.

The analysis of the zinc recovered in the zinc circuit showed that the organic depressant DS-20 and pH both had an affect on zinc metallurgy. The parameter of DS-20 had the greatest affect, with the use of the reagent giving much better zinc recoveries. The parameter of pH also had an affect, with the high pH values giving poorer zinc metallurgy. The overall recoveries are still somewhat low, and therefore, more test work is needed on this system.

Author(s)	technical notes	Project No.	Page	of
D. Taylor			4	5

Discussion:

TEST NUMBER	DS-20 G/TONNE	ZINC RECOVERED IN 2ND ZN CONC.
C-33	0	53.70
C-34	30	71.24
C-35	0	37.65
C-36	30	57.10

TEST NUMBER	PH	ZINC RECOVERED IN 2ND ZN CONC.
C-33	10.0	53.70
C-34	10.0	71.24
C-35	11.5	37.65
C-36	11.5	57.10

Method: A fractional factorial design for the lead circuit was set up using the following parameters:

TEST NUMBER	COLLECTOR AT 50 G/TONNE	DS-20 G/TONNE	PH
C-33	Xanthate	0	8.5
C-34	R-242	0	9.5
C-35	Xanthate	30	9.5
C-36	R-242	30	8.5

For the zinc circuit:

TEST NUMBER	DS-20 G/TONNE	PH	COPPER SULPHATE G/TONNE
C-33	0	10.0	700
C-34	30	10.0	1000
C-35	0	11.5	1000
C-36	30	11.5	700

The tests were run by grinding approximately 2000 g of ore in the laboratory rod mill with 1000 mL of water, 150 g/tonne SO_2, and 2 kg/tonne sodium carbonate. The laboratory mill was run for three hours to produce the desired grind. The sample was removed from the mill and split by a bucket-mixer-beaker method into four approximately equal samples.

The samples were then placed in a 500 g stainless steel flotation cell, then placed on the Denver laboratory flotation machine. The tests were then run according to the flotation test reports (shown in figures 1 to 4). The resultant flotation samples were filtered, dried and assayed. The data was then placed on a standard metallurgical balance sheet (shown in figures 5 to 8). The results from the metallurgical balances were then used in the fractional factorial design system to produce the coefficient of variance data, and to show the important parameters in the test.

FIGURE 1

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE:	Caribou	**PROJECT NUMBER:**	H-76815
SAMPLE WEIGHT:	500 g	**TEST NUMBER:**	C-33
PARTICLE SIZE:		**TEST BY:**	
PURPOSE:	Preliminary Factorial Design "Pb Float"	**DATE:**	03/10/88
BACKGROUND:		**CLIENT:**	

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)						Xanthate no DS20 pH 8.5
					DS20	Xanthate	MIBC				
Aeration	20.0			1400 full							
pH			soda ash 8.5								
DS20					-						
Condition	5.0			1400 none							
Xanthate						2.5 g					
Condition	5.0			1400 none							
Float 1	1.0			2100			2 drops				
Float 2	2.0										
Float 3	4.0										

COMMENTS:

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH

P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Caribou
SAMPLE WEIGHT: 500 g
PARTICLE SIZE:
PURPOSE: Preliminary Factorial Design "Zn Float"
BACKGROUND:

PROJECT NUMBER: H-76815
TEST NUMBER: C-33
TEST BY:
DATE: 03/10/88
CLIENT:

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)						DS20 - 0.0 pH 10.0 CuSO4 - 700
					DS20	CuSO4	Xanthate	MIBC			
pH			Lime 10.0	1400 none							
DS20	5.0			1400 none	-						
pH & CuSO4						3.5 g					
Condition	5.0			1400 none							
Xanthate							3.5 g				
Condition	5.0			1400 none				2 drops			
Float 1	1.0			2100							
Float 2	2.0										
Float 3	4.0										

COMMENTS:

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Caribou
SAMPLE WEIGHT: 500 g
PARTICLE SIZE:
PURPOSE: Preliminary Factorial Design "Pb Float"
BACKGROUND:

PROJECT NUMBER: H-76815
TEST NUMBER: C-34
TEST BY:
DATE: 03/10/88
CLIENT:

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)						242 no D820 pH 9.5
					D820	Xanthate	MIBC				
Aeration	20.0			1400 full							
pH			soda ash 9.5								
D820					-						
Condition	5.0			1400 none							
242						2.5 g					
Condition	5.0			1400 none							
Float 1	1.0			2100			2 drops				
Float 2	2.0										
Float 3	4.0										

COMMENTS:

FIGURE 2

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Caribou
SAMPLE WEIGHT: 500 g
PARTICLE SIZE:
PURPOSE: Preliminary Factorial Design "Zn Float"
BACKGROUND:

PROJECT NUMBER: H-76815
TEST NUMBER: C-34
TEST BY:
DATE: 03/10/88
CLIENT:

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)						D820- 30 g/t pH 10.0 CuSO4-1000 g/t
					D820	CuSO4	Xanthate	MIBC			
pH			Lime 10.0	1400 none							
D820	5.0				1.5 g						
pH & CuSO4						5.0 g					
Condition	5.0			1400 none							
Xanthate							3.5 g				
Condition	5.0			1400 none				2 drops			
Float 1	1.0			2100							
Float 2	2.0										
Float 3	4.0										

COMMENTS:

FIGURE 3

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Caribou
SAMPLE WEIGHT: 500 g
PARTICLE SIZE:
PURPOSE: Preliminary Factorial Design "Pb Float"
BACKGROUND:

PROJECT NUMBER: H-76815
TEST NUMBER: C-35
TEST BY:
DATE: 03/10/88
CLIENT:

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)						Xanthate DS20- 30 g/t pH 9.5
					DS20	Xanthate	MIBC				
Aeration	20.0			1400 full							
pH			soda ash 9.5								
DS20					1.5						
Condition	5.0			1400 none							
Xanthate						2.5 g					
Condition	5.0			1400 none							
Float 1	1.0			2100			2 drops				
Float 2	2.0										
Float 3	4.0										

COMMENTS:

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE:	Caribou	PROJECT NUMBER:	H-76815
SAMPLE WEIGHT:	500 g	TEST NUMBER:	C-35
PARTICLE SIZE:		TEST BY:	
PURPOSE:	Preliminary Factorial Design "En Float"	DATE:	03/10/88
BACKGROUND:		CLIENT:	

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)						DS20- 30 g/t pH 10.0 CuSO4-1000 g/t
					DS20	CuSO4	Xanthate	MIBC			
pH			Lime 11.5	1400 none							
DS20					-						
pH & CuSO4						5.0 g					
Condition	5.0			1400 none							
Xanthate							3.5 g				
Condition	5.0			1400 none				2 drops			
Float 1	1.0										
Float 2	2.0										
Float 3	4.0										

COMMENTS:

FIGURE 4

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB. E3B 5H1, (506) 453-2206

SAMPLE: Caribou
SAMPLE WEIGHT: 500 g
PARTICLE SIZE:
PURPOSE: Preliminary Factorial Design "Pb Float"
BACKGROUND:

PROJECT NUMBER: H-76815
TEST NUMBER: C-36
TEST BY:
DATE: 03/10/88
CLIENT:

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)						242 D820- 30 g/t pH 8.5
					D820	Xanthate	MIBC				
Aeration	20.0			1400 full							
pH			soda ash 8.5								
D820					1.5 g						
Condition	5.0			1400 none							
242						2.5 g					
Condition	5.0			1400 none							
Float 1	1.0			2100			2 drops				
Float 2	2.0										
Float 3	4.0										

COMMENTS:

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Caribou
SAMPLE WEIGHT: 500 g
PARTICLE SIZE:
PURPOSE: Preliminary Factorial Design "Zn Float"
BACKGROUND:

PROJECT NUMBER: H-76815
TEST NUMBER: C-36
TEST BY:
DATE: 03/10/88
CLIENT:

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)						DS20- 30 g/t pH 11.5 CuSO4-700 g/t
					DS20	CuSO4	Xanthate	MIBC			
pH			Lime 11.5	1400 none							
DS20	5.0				1.5 g						
pH & CuSO4						3.5 g					
Condition	5.0										
Xanthate							3.5 g				
Condition	5.0							2 drops			
Float 1	1.0										
Float 2	2.0										
Float 3	4.0										

COMMENTS:

FIGURE 5

CARIBOU FLOTATION TEST C-33

METAL DISTRIBUITIONS

DATE : OCTOBER 12,1988

SAMPLE NAME	SAMPLE WEIGHT (grams)	Wt. Percent	METAL Pb (%)	METAL Zn (%)	Pb DIST (%)	Zn DIST (%)	CUMUL Pb DIST	CUMUL Zn DIST
Pb CONC 1	81.4	26.01	9.33	8.04	48.46	23.95	48.46	23.95
Pb CONC 2	19.2	6.15	10.20	9.85	12.52	6.93	60.98	30.88
Pb CONC 3	11.4	3.64	8.34	10.90	6.06	4.55	67.04	35.43
Zn CONC 1	24.8	7.93	3.56	23.80	5.64	21.62	72.68	57.04
Zn CONC 2	12.3	3.93	4.94	29.00	3.88	13.06	76.56	70.10
Zn CONC 3	6.6	2.11	5.89	22.10	2.48	5.34	79.04	75.44
TAILS	157.2	50.22	2.09	4.27	20.96	24.6	100.00	100.00
	0.0	0.00	0.00	0.00	0.00	ERR	100.00	ERR
	313.1							

METAL CONTENT (Pb)	METAL CONTENT (Zn)	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL GRADE (Pb&Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
242.71	209.15	9.33	8.04	17.37	48.46	23.95
62.69	60.54	9.50	8.39	17.88	60.98	30.88
30.37	39.69	9.38	8.64	18.02	67.04	35.43
28.24	188.77	8.32	11.39	19.71	72.68	57.04
19.42	114.03	8.04	12.84	20.89	76.56	70.10
12.44	46.66	7.95	13.24	21.19	79.04	75.44
104.97	214.45	5.01	8.73	13.74	100.00	100.00
0.00	0.00	5.01	8.73	13.74	100.00	ERR
500.83	873.30					

ZINC CIRCUIT

	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
Zn CONC 1	3.56	23.80	17.11	33.47
Zn CONC 2	4.02	25.52	28.87	53.70
Zn CONC 3	4.30	25.01	36.41	61.97
TAILS	2.57	8.78	100.00	100.00

FIGURE 6

CARIBOU FLOTATION TEST C-34

METAL DISTRIBUITIONS

DATE : OCTOBER 12,1988

SAMPLE NAME	SAMPLE WEIGHT (grams)	Wt. Percent	METAL Pb (%)	METAL Zn (%)	Pb DIST (%)	Zn DIST (%)	CUMUL Pb DIST	CUMUL Zn DIST
Pb CONC 1	134.7	30.22	10.00	8.12	58.61	28.05	58.61	28.05
Pb CONC 2	28.5	6.40	9.43	10.00	11.70	7.31	70.31	35.36
Pb CONC 3	17.8	3.99	7.28	11.40	5.64	5.20	75.95	40.56
Zn CONC 1	45.8	10.27	2.10	27.40	4.18	32.18	80.13	72.74
Zn CONC 2	18.0	4.04	3.23	22.00	2.53	10.17	82.67	82.91
Zn CONC 3	8.7	1.95	13.30	11.00	5.04	2.46	87.71	85.36
TAILS	192.2	43.12	1.47	2.97	12.29	14.6	100.00	100.00
	0.0	0.00	0.00	0.00	0.00	ERR	100.00	ERR
	445.8							

METAL CONTENT (Pb)	METAL CONTENT (Zn)	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL GRADE (Pb&Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
302.20	245.39	10.00	8.12	18.12	58.61	28.05
60.33	63.98	9.90	8.45	18.35	70.31	35.36
29.07	45.52	9.64	8.74	18.38	75.95	40.56
21.58	281.52	8.12	12.51	20.63	80.13	72.74
13.06	88.94	7.76	13.21	20.97	82.67	82.91
25.99	21.49	7.96	13.13	21.08	87.71	85.36
63.38	128.06	5.16	8.75	13.91	100.00	100.00
0.00	0.00	5.16	8.75	13.91	100.00	ERR
515.61	874.90					

ZINC CIRCUIT

	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
Zn CONC 1	2.10	27.40	17.40	54.14
Zn CONC 2	2.42	25.88	27.93	71.24
Zn CONC 3	3.73	24.09	48.89	75.37
TAILS	2.09	8.76	100.00	100.00

FIGURE 7

CARIBOU FLOTATION TEST C-35

METAL DISTRIBUITIONS

DATE : OCTOBER 12,1988

SAMPLE NAME	SAMPLE WEIGHT (grams)	Wt. Percent	METAL Pb (%)	METAL Zn (%)	Pb DIST (%)	Zn DIST (%)	CUMUL Pb DIST	CUMUL Zn DIST
Pb CONC 1	110.4	24.98	10.90	8.71	55.29	24.38	55.29	24.38
Pb CONC 2	18.1	4.09	11.80	10.60	9.81	4.86	65.09	29.24
Pb CONC 3	18.5	4.19	8.04	11.90	6.84	5.59	71.94	34.84
Zn CONC 1	38.2	8.65	2.69	16.80	4.72	16.28	76.66	51.11
Zn CONC 2	11.8	2.68	4.01	27.50	2.18	8.26	78.84	59.37
Zn CONC 3	13.8	3.13	2.90	34.40	1.84	12.05	80.68	71.41
TAILS	231.0	52.28	1.82	4.88	19.32	28.6	100.00	100.00
	0.0	0.00	0.00	0.00	0.00	ERR	100.00	ERR
	441.9							

METAL CONTENT (Pb)	METAL CONTENT (Zn)	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL GRADE (Pb&Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
272.31	217.60	10.90	8.71	19.61	55.29	24.38
48.30	43.39	11.03	8.98	20.00	65.09	29.24
33.71	49.90	10.65	9.34	19.99	71.94	34.84
23.26	145.26	9.01	10.88	19.89	76.66	51.11
10.74	73.68	8.71	11.88	20.59	78.84	59.37
9.06	107.50	8.33	13.36	21.68	80.68	71.41
95.15	255.13	4.93	8.92	13.85	100.00	100.00
0.00	0.00	4.93	8.92	13.85	100.00	ERR
492.54	892.46					

ZINC CIRCUIT

	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
Zn CONC 1	2.69	16.80	16.83	24.98
Zn CONC 2	3.00	19.33	24.60	37.65
Zn CONC 3	2.98	22.59	31.16	56.13
TAILS	2.07	8.72	100.00	100.00

FIGURE 8

CARIBOU FLOTATION TEST C-36

METAL DISTRIBUITIONS

DATE : OCTOBER 12,1988

SAMPLE NAME	SAMPLE WEIGHT (grams)	Wt. Percent	METAL Pb (%)	METAL Zn (%)	Pb DIST (%)	Zn DIST (%)	CUMUL Pb DIST	CUMUL Zn DIST
Pb CONC 1	136.1	38.70	6.79	6.92	51.80	30.12	51.80	30.12
Pb CONC 2	14.1	4.01	9.60	6.09	7.60	2.75	59.40	32.87
Pb CONC 3	15.2	4.32	9.37	7.08	7.98	3.44	67.37	36.31
Zn CONC 1	38.0	10.82	2.82	21.80	6.01	26.52	73.39	62.83
Zn CONC 2	15.9	4.51	3.66	19.40	3.26	9.85	76.64	72.68
Zn CONC 3	7.9	2.24	4.53	18.80	2.00	4.74	78.64	77.42
TAILS	124.5	35.40	3.06	5.67	21.36	22.6	100.00	100.00
	0.0	0.00	0.00	0.00	0.00	ERR	100.00	ERR
	351.7							

METAL CONTENT (Pb)	METAL CONTENT (Zn)	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL GRADE (Pb&Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
262.74	267.77	6.79	6.92	13.71	51.80	30.12
38.54	24.45	7.05	6.84	13.90	59.40	32.87
40.47	30.58	7.27	6.86	14.13	67.37	36.31
30.50	235.78	6.44	9.66	16.09	73.39	62.83
16.51	87.53	6.23	10.36	16.60	76.64	72.68
10.16	42.17	6.18	10.65	16.83	78.64	77.42
108.32	200.72	5.07	8.89	13.96	100.00	100.00
0.00	0.00	5.07	8.89	13.96	100.00	ERR
507.25	889.00					

ZINC CIRCUIT

	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
Zn CONC 1	2.82	21.80	18.43	41.64
Zn CONC 2	3.07	21.09	28.41	57.10
Zn CONC 3	3.25	20.80	34.55	64.55
TAILS	3.12	10.69	100.00	100.00

APPENDIX 14

Caribou Factorial Design Flotation Test
C-37 to C-40

PROCESS STUDIES GROUP

technical notes		Project No.	File No.
Project Flotation		Date 10/02/89	Page 1

Subject: Caribou Factorial Design Flotation Test C-37 to C-40

Author(s): Donald S. Taylor

Objective: With the completion of the first five tests on Caribou ore, it was determined that procedures must be found to lower the amount of zinc recovered in the lead concentrate without lowering the metallurgical performance of the lead circuit. In a continuing effort to improve the test work results, we contacted East-West Caribou Resources Ltd., and explained our difficulty in obtaining adequate metallurgical performance from their ore. The company representatives could not see any problems with our test procedures, and at their suggestion, the flotation data was sent to them for evaluation. The company examined the data and suggested that our flotation cell agitating speed was too fast. With this information, we constructed a new factorial design test, in which we could compare the results of slower agitation with previously run tests at higher agitation speeds.

Conclusions: The tests showed that the higher agitation speeds produced better overall lead metallurgy. The lead recovered was on the average 14.29% higher in the faster speed agitator tests. However, the zinc recovered in the high speed agitator tests was 10.8% higher than with slow speed agitation. The slow speed agitation tests, therefore, did not produce the improved metallurgical performance expected.

The tests on the lead circuit showed that aeration, xanthate addition rates and pH all had an affect on lead metallurgy. The use of natural aeration at normal pH values produced better results than using aeration with SO_2 at pH 6.0. The xanthate parameter indicated that higher recoveries occurred with larger quantities of xanthate, and higher pH values generally produced better lead recoveries. The zinc recovered in the lead circuit continued to be high in the 26-40% zinc recovery range. In one test, the zinc recovered in the lead concentrate was only 9.06%, but the lead recovery in that test was very low at 18.4%, compared to the normal expected recovery of about 75% lead.

The tests in the zinc circuit showed that pH and copper sulphate had an affect on zinc metallurgy. The higher pH values produced poorer zinc recoveries, while the use of lower quantities of copper sulphate produced better zinc recoveries. The zinc recoveries, however, are still low, and more test work is needed on the flotation circuits.

Discussion: The results of the first five tests completed on Caribou ore showed flotation results that had poor lead metallurgy and high recoveries of zinc in the lead concentrate. In an effort to overcome these flotation difficulties, East-West Caribou Mining was contacted to determine if they could help us in our laboratory testing program. At their suggestion, we sent to them the data from tests C-17 to C-32 for their evaluation. The company examined the data and concluded that our reagent schemes seemed all right, our test procedures were adequate, but our lead grade and recovery were poor. They compared our data to laboratory tests done in Kamloops, B.C. and found our recovery data occurred at the far (poor) end of their grade/recovery curve. The only conclusion they could come to about the testing was that our agitation speed during flotation was too fast. As a result of their study, we concluded a test of slower agitator speeds was essential. A series of tests would, therefore, be conducted at slower agitator speeds (1600 versus 2100 rpm) to determine if this parameter could produce better metallurgy.

In conjunction with the agitator speed tests, it was determined a full factorial design test on both lead and zinc would be carried out. The parameters chosen for the lead test would be aeration (both natural and at a pH of 6.0 with SO_2 gas), pH, and xanthate addition levels. These parameters were chosen so the tests would have a minimum of reagents, and the results could be compared to previously run tests at higher agitator speeds. The levels of reagents were xanthate at 50 and 100 g/tonne, pH at 9.3 and 9.8, and aeration with and without SO_2 gas. These parameters were used for the lead circuit fractional factorial design.

The previous set of tests run on the zinc circuit showed that the zinc metallurgy was poor, especially with low zinc recoveries. It was determined also in this case to use simple reagent schemes to determine if the agitator speed parameters had an affect on zinc metallurgy. The factors for the factorial design were chosen as pH at 10.2 and 11.0, copper sulphate at 700 and 1000 g/tonne, and xanthate at 70 and 100 g/tonne.

The test's primary objective was to determine if agitator speed had an effect on metallurgy. Therefore, we ran this complete series of tests at the slower agitator speed, so they could be compared to earlier run tests of the same reagent scheme at higher agitator speeds. The results of the lower agitator speeds produced lead recoveries on the average 14.2% lower than with higher speeds. However, lower agitator speeds lowered the zinc recovery 10.8% in the lead concentrate on the average. The results in the zinc circuit indicated that the slower agitator speeds would lower the zinc recovery on the conclusion that lower agitator speeds do not improve overall metallurgy.

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Discussion:

TEST NUMBER	AGITATOR SPEED RPM	LEAD RECOVERED IN 2ND PB CONC.	ZINC RECOVERED IN 2ND PB CONC.
C-22	2100	75.04	42.10
C-39	1600	56.68	26.43
C-26	2100	75.96	46.80
C-37	1600	66.00	40.79

TEST NUMBER	AGITATOR SPEED RPM	ZINC RECOVERED IN 2ND ZN CONC.
C-19	2100	90.78
C-39	1600	78.40
C-20	2100	93.63
C-40	1600	52.44

A fractional factorial design was constructed, and the test work completed. The data on the recovery of minerals in the second concentrate fraction of each test was used in the mathematical analysis. The F test distribution was used for the analysis of the test results, with F for the test design being set at 7.71. The analysis of the lead circuit showed that all parameters tested were important.

The most important parameter in the flotation test for lead was the aeration step, where natural aeration was found to improve lead recovery. The second most important parameter was the addition of xanthate, with the higher additions of xanthate producing better lead recoveries. The pH parameter had a very small effect on lead metallurgy, but higher pH values produced slightly better recoveries on the average.

Discussion:

TEST NUMBER	AERATION	LEAD RECOVERY IN 2ND PB CONC	ZINC RECOVERY IN 2ND PB CONC
C-37	natural	66.00	40.79
C-38	6.0 pH with SO_2	18.39	9.06
C-39	natural	56.68	26.43
C-40	6.0 pH with SO_2	38.45	28.94

TEST NUMBER	XANTHATE G/TONNE	LEAD RECOVERY IN 2ND PB CONC	ZINC RECOVERY IN 2ND PB CONC
C-37	100	66.00	40.79
C-38	50	18.39	9.06
C-39	50	56.68	26.43
C-40	100	38.45	28.94

TEST NUMBER	PH	LEAD RECOVERY IN 2ND PB CONC	ZINC RECOVERY IN 2ND PB CONC
C-37	9.3	66.00	40.79
C-38	9.3	18.39	9.06
C-39	9.8	56.68	26.43
C-40	9.8	38.45	28.94

Discussion:

The zinc recovered in the lead circuit was affected by the xanthate and aeration parameters of the factorial design. If the flotation test is conducted at higher xanthate levels, there is more zinc recovered in the lead concentrate. However using the lower pH of 6.0 by SO_2 gas in the aeration step, produces on the average lower zinc recoveries in the lead circuit. One test result showed a zinc recovery of only 9.06% zinc in the lead circuit. This result for zinc is good, but the lead recovery was only 18.39% lead, and this is very poor compared to our expectation for lead recovery of 75% lead. Therefore, the zinc results for this test are not normal for this ore.

The test results of the lead circuit continue to cause problems in improving overall metallurgy. If we operate with natural aeration, we produce better lead metallurgy, which is beneficial, but also increased zinc metallurgy, which is detrimental. If we add higher additions of xanthate, we improve the lead recovery, which is beneficial, but also improve the zinc metallurgy, which is detrimental. The contradictory nature of the lead and zinc in the lead concentrate continues to occur, therefore, more test work is needed.

The analysis of the zinc recovered in the zinc circuit showed that pH and copper sulphate both had an effect on zinc metallurgy. The parameter of pH had the greatest effect with lower pH values producing much better metallurgy. The parameter of copper sulphate also had an effect with higher additions of copper sulphate producing better zinc metallurgy. The overall recoveries are still low, therefore, more test work is needed in this system.

TEST NUMBER	PH	ZINC RECOVERED IN 2ND ZN CONC.
C-37	10.2	67.38
C-38	11.0	38.29
C-39	10.2	78.40
C-40	11.0	52.44

Author(s)	technical notes	Project No.	Page of
D. Taylor			6 7

Discussion:

TEST NUMBER	COPPER SULPHATE G/TONNE	ZINC RECOVERY IN 2ND ZINC CONC.
C-37	700	67.38
C-38	700	38.29
C-39	1000	78.40
C-40	1000	52.44

Method: A fractional factorial design for the lead circuit was set up using the following parameters:

TEST NUMBER	AERATION	PH	XANTHATE G/TONNE
C-37	natural	9.3	100
C-38	6.0 pH with SO2	9.3	50
C-39	natural	9.8	50
C-40	6.0 pH with SO2	9.8	100

For the zinc circuit:

TEST NUMBER	PH	COPPER SULPHATE G/TONNE	XANTHATE G/TONNE
C-37	10.2	700	100
C-38	11.0	700	70
C-39	10.2	1000	70
C-40	11.0	1000	100

The tests were run by grinding approximately 2000 g of ore in the laboratory rod mill with 1000 mL of water, 150 g/tonne SO_2, and 2 kg/tonne sodium carbonate. The laboratory mill was run for three hours to produce the desired grind. The sample was removed from the mill and split by a bucket-mixer-beaker method into four approximately equal samples.

The samples were then placed in a 500 g stainless steel flotation cell, then placed on the Denver laboratory flotation machine. The tests were then run according to the flotation test reports (shown in figures 1 to 4). The resultant flotation samples were filtered, dried and assayed. The data was then placed on a standard metallurgical balance sheet (shown in figures 5 to 8). The results from the metallurgical balances were then used in the fractional factorial design system to produce the coefficient of variance data, and to show the important parameters in the test.

FIGURE 1

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Caribou
SAMPLE WEIGHT: 500 g
PARTICLE SIZE:
PURPOSE: Preliminary Factorial Design Test "Pb Float"
BACKGROUND:

PROJECT NUMBER: H-76815
TEST NUMBER: C-37
TEST BY: D. Taylor
DATE:
CLIENT:

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					aeration - natural pH - 9.3 Xanthate - 100 g/t
					Xanthate	MIBC				
Aeration	20.0			1400 full						
pH adjust			soda ash 9.3							
Xanthate					5 g @ 1%					
Condition	5.0			1400 off						
Float 1	1.0			1600		2 drops				
Float 2	2.0									
Float 3	4.0									

COMMENTS:

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Caribou
SAMPLE WEIGHT: 500 g
PARTICLE SIZE:
PURPOSE: Preliminary Factorial Design Test "Zn Float"
BACKGROUND:

PROJECT NUMBER: H-76815
TEST NUMBER: C-37
TEST BY: D. Taylor
DATE:
CLIENT:

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					pH - 10.2 CuSO4 - 700 g/t Xanthate - 100 g/t
					CuSO4	Xanthate	MIBC			
pH & CuSO4			Lime 10.2							
Condition	10.0			1400 none	3.5 g					
Xanthate						5.0 g				
Condition	5.0			1400 none						
Float 1	1.0			1600			2 drops			
Float 2	2.0									
Float 3	4.0									

COMMENTS:

FIGURE 2

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Caribou
SAMPLE WEIGHT: 500 g
PARTICLE SIZE:
PURPOSE: Preliminary Factorial Design Test "Pb Float"
BACKGROUND:

PROJECT NUMBER: H-76815
TEST NUMBER: C-38
TEST BY: D. Taylor
DATE:
CLIENT:

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					aeration - SO2 pH 6.0 pH - 9.3 Xanthate - 50 g/t
					Xanthate	MIBC				
Aeration	20.0		SO2 pH 6.0	1400 full						
pH adjust			soda ash 9.3							
Xanthate					2.5 g					
Condition	5.0			1400 off						
Float 1	1.0			1600		2 drops				
Float 2	2.0									
Float 3	4.0									

COMMENTS:

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Caribou
SAMPLE WEIGHT: 500 g
PARTICLE SIZE:
PURPOSE: Preliminary Factorial Design Test "Zn Float"
BACKGROUND:

PROJECT NUMBER: H-76815
TEST NUMBER: C-38
TEST BY: D. Taylor
DATE:
CLIENT:

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					pH - 11.0 CuSO4 - 700 g/t Xanthate - 70 g/t
					CuSO4	Xanthate	MIBC			
pH & CuSO4			Lime 11.0							
Condition	10.0			1400 none	3.5 g					
Xanthate						3.5 g				
Condition	5.0			1400 none						
Float 1	1.0			1600			2 drops			
Float 2	2.0									
Float 3	4.0									

COMMENTS:

FIGURE 3

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Caribou
SAMPLE WEIGHT: 500 g
PARTICLE SIZE:
PURPOSE: Preliminary Factorial Design Test "Pb Float"
BACKGROUND:

PROJECT NUMBER: H-76815
TEST NUMBER: C-39
TEST BY: D. Taylor
DATE:
CLIENT:

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					aeration - natural pH - 9.8 Xanthate - 50 g/t
					Xanthate	MIBC				
Aeration	20.0			1400 full						
pH adjust			soda ash 9.8							
Xanthate					2.5 g					
Condition	5.0			1400 off						
Float 1	1.0			1600		2 drops				
Float 2	2.0									
Float 3	4.0									

COMMENTS:

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH

P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE:	Caribou	PROJECT NUMBER:	H-76815
SAMPLE WEIGHT:	500 g	TEST NUMBER:	C-38
PARTICLE SIZE:		TEST BY:	D. Taylor
PURPOSE:	Preliminary Factorial Design Test "Zn Float"	DATE:	
BACKGROUND:		CLIENT:	

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					pH - 10.2 CuSO4 - 1000 g/t Xanthate - 70 g/t
					CuSO4	Xanthate	MIBC			
pH & CuSO4			Lime 10.2		5.0 g					
Condition	10.0			1400 none						
Xanthate						3.5 g				
Condition				1400 none						
Float 1	1.0			1600			2 drops			
Float 2	2.0									
Float 3	4.0									

COMMENTS:

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE: Caribou
SAMPLE WEIGHT: 500 g
PARTICLE SIZE:
PURPOSE: Preliminary Factorial Design Test "Pb Float"
BACKGROUND:

PROJECT NUMBER: H-76815
TEST NUMBER: C-40
TEST BY: D. Taylor
DATE:
CLIENT:

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					aeration - 6.0 with SO2 pH - 9.8 Xanthate - 100 g/t
					Xanthate	MIBC				
Aeration	20.0		SO2 pH 6.0	1400 full						
pH adjust			soda ash 9.8							
Xanthate					5.0 g					
Condition	5.0			1400 none						
Float 1	1.0			1600		2 drops				
Float 2	2.0									
Float 3	4.0									

COMMENTS:

FIGURE 4

FLOTATION TEST REPORT - MINERAL DEVELOPMENT BRANCH
P.O. BOX 6000, FREDERICTON, NB E3B 5H1, (506) 453-2206

SAMPLE:	Caribou	**PROJECT NUMBER:**	H-76815
SAMPLE WEIGHT:	500 g	**TEST NUMBER:**	C-40
PARTICLE SIZE:		**TEST BY:**	D. Taylor
PURPOSE:	Preliminary Factorial Design Test "Zn Float"	**DATE:**	
BACKGROUND:		**CLIENT:**	

OPERATION	TIME (MIN)	% SOLIDS SOLID SP.GR.	ACIDITY (PH)	AGITATION (RPM) AIR (CFM)	REAGENTS (KG/TONNE)					pH - 11.0 CuSO4 - 1000 g/t Xanthate - 100 g/t
					CuSO4	Xanthate	MIBC			
pH & CuSO4			Lime 11.2		5.0 g					
Condition	10.0			1400 none						
Xanthate						5.0 g				
Condition	5.0			1400 none						
Float 1	1.0			1600			2 drops			
Float 2	2.0									
Float 3	4.0									

COMMENTS:

FIGURE 5

CARIBOU FLOTATION TEST C-37

METAL DISTRIBUITIONS

DATE : DECEMBER 28,1988

SAMPLE NAME	SAMPLE WEIGHT (grams)	Wt. Percent	METAL Pb (%)	METAL Zn (%)	Pb DIST (%)	Zn DIST (%)	CUMUL Pb DIST	CUMUL Zn DIST
Pb CONC 1	168.5	35.66	7.93	7.54	54.87	31.48	54.87	31.48
Pb CONC 2	40.9	8.66	6.54	9.30	11.13	9.31	66.00	40.79
Pb CONC 3	22.9	4.85	5.41	10.40	5.16	5.83	71.16	46.62
Zn CONC 1	37.2	7.88	2.45	34.10	3.79	31.05	74.95	77.67
Zn CONC 2	8.6	1.81	3.82	23.50	1.36	4.92	76.32	82.59
Zn CONC 3	10.1	2.13	4.75	20.40	1.99	5.02	78.31	87.60
TAILS	184.4	39.01	2.83	2.75	21.69	12.4	100.00	100.00
	0.0	0.00	0.00	0.00	0.00	ERR	100.00	ERR
	472.5							

METAL CONTENT (Pb)	METAL CONTENT (Zn)	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL GRADE (Pb&Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
279.20	272.43	7.93	7.54	15.47	54.87	31.48
56.65	80.56	7.58	7.96	15.54	66.00	40.79
26.36	50.48	7.36	8.20	15.57	71.16	46.62
19.30	268.68	6.69	11.78	18.47	74.95	77.67
6.92	42.56	6.60	12.14	18.74	76.32	82.59
10.11	43.42	6.53	12.43	18.96	78.31	87.60
110.39	107.27	5.09	8.65	13.74	100.00	100.00
0.00	0.00	5.09	8.65	13.74	100.00	ERR
508.93	865.40					

ZINC CIRCUIT

	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
Zn CONC 1	2.45	34.10	13.16	58.16
Zn CONC 2	2.71	32.12	17.87	67.38
Zn CONC 3	3.07	30.01	24.75	75.78
TAILS	2.93	9.09	100.00	100.00

FIGURE 6

CARIBOU FLOTATION TEST C-38

METAL DISTRIBUITIONS DATE : DECEMBER 28,1988

SAMPLE NAME	SAMPLE WEIGHT (grams)	Wt. Percent	METAL Pb (%)	METAL Zn (%)	Pb DIST (%)	Zn DIST (%)	CUMUL Pb DIST	CUMUL Zn DIST
Pb CONC 1	54.8	11.00	7.13	6.16	15.17	7.79	15.17	7.79
Pb CONC 2	10.4	2.09	7.97	5.29	3.22	1.27	18.39	9.06
Pb CONC 3	7.6	1.53	9.45	6.90	2.79	1.21	21.18	10.27
Zn CONC 1	41.6	8.37	4.88	20.90	7.89	20.09	29.07	30.36
Zn CONC 2	23.1	4.55	5.40	26.70	4.85	14.27	33.93	44.63
Zn CONC 3	26.0	5.22	6.43	24.80	6.49	14.87	40.41	59.50
TAILS	334.1	67.14	4.59	5.25	59.59	40.5	100.00	100.00
	0.0	0.00	0.00	0.00	0.00	ERR	100.00	ERR
	497.6							

METAL CONTENT (Pb)	METAL CONTENT (Zn)	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL GRADE (Pb&Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
78.46	67.79	7.13	6.16	13.29	15.17	7.79
16.64	11.04	7.25	6.02	13.27	18.39	9.06
14.43	10.54	7.49	6.11	13.61	21.18	10.27
40.83	174.88	6.54	11.50	18.04	29.07	30.36
25.11	124.16	6.35	14.05	20.40	33.93	44.63
33.56	129.42	6.36	15.76	22.12	40.41	59.50
308.19	352.51	5.17	8.70	13.88	100.00	100.00
0.00	0.00	5.17	8.70	13.88	100.00	ERR
517.23	870.34					

ZINC CIRCUIT

	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
Zn CONC 1	4.88	20.90	10.02	22.39
Zn CONC 2	5.07	22.97	16.18	38.29
Zn CONC 3	5.46	23.49	24.41	54.86
TAILS	4.78	9.15	100.00	100.00

FIGURE 7

CARIBOU FLOTATION TEST C-39

METAL DISTRIBUITIONS   DATE : DECEMBER 28,1988

SAMPLE NAME	SAMPLE WEIGHT (grams)	Wt. Percent	METAL Pb (%)	METAL Zn (%)	Pb DIST (%)	Zn DIST (%)	CUMUL Pb DIST	CUMUL Zn DIST
Pb CONC 1	129.4	21.69	9.70	7.73	47.58	21.91	47.58	21.91
Pb CONC 2	24.2	4.61	9.94	8.55	9.10	4.52	56.68	26.43
Pb CONC 3	15.9	3.02	9.89	9.31	5.94	3.44	62.62	29.87
Zn CONC 1	77.4	14.76	3.25	27.90	9.56	47.08	72.17	76.95
Zn CONC 2	24.1	4.59	4.68	15.00	4.27	7.30	76.44	84.85
Zn CONC 3	21.7	4.14	5.38	10.00	4.43	4.76	80.87	89.61
TAILS	231.5	44.19	2.18	2.05	19.13	10.4	100.00	100.00
	0.0	0.00	0.00	0.00	0.00	ERR	100.00	ERR
	524.2							

METAL CONTENT (Pb)	METAL CONTENT (Zn)	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL GRADE (Pb&Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
239.54	190.89	9.70	7.73	17.43	47.58	21.91
45.80	39.39	9.74	7.96	17.60	56.68	26.43
29.91	29.97	9.75	8.05	17.80	62.62	29.87
48.11	410.30	7.72	14.24	21.96	72.17	76.95
21.43	68.85	7.45	14.31	21.76	76.44	84.55
22.29	41.44	7.29	13.99	21.28	80.87	89.61
96.31	90.57	5.03	8.71	13.75	100.00	100.00
0.00	0.00	5.03	8.71	13.75	100.00	ERR
503.45	871.41					

ZINC CIRCUIT

	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
Zn CONC 1	3.26	27.90	26.56	57.13
Zn CONC 2	3.80	24.75	36.39	78.40
Zn CONC 3	3.91	22.16	48.32	85.18
TAILS	2.79	9.03	100.00	100.00

FIGURE 8

CARIBOU FLOTATION TEST C-40

METAL DISTRIBUITIONS DATE : DECEMBER 28, 1988

SAMPLE NAME	SAMPLE WEIGHT (grams)	Wt. Percent	METAL Pb (%)	METAL Zn (%)	Pb DIST (%)	Zn DIST (%)	CUMUL Pb DIST	CUMUL Zn DIST
Pb CONC 1	139.8	26.34	6.07	7.55	32.28	23.14	32.28	23.14
Pb CONC 2	28.4	5.35	5.71	9.31	6.17	5.80	38.45	28.94
Pb CONC 3	10.4	1.95	5.70	11.70	2.25	2.66	40.70	31.60
Zn CONC 1	83.6	15.75	4.24	14.30	13.49	26.22	54.19	57.82
Zn CONC 2	21.7	4.08	5.31	20.30	4.38	9.65	58.57	67.47
Zn CONC 3	19.5	3.67	6.07	15.70	4.50	6.70	63.06	74.17
TAILS	227.5	42.84	4.27	5.18	35.94	25.8	100.00	100.00
	0.0	0.00	0.00	0.00	0.00	ERR	100.00	ERR
	530.9							

METAL CONTENT (Pb)	METAL CONTENT (Zn)	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL GRADE (Pb&Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
159.89	198.88	6.07	7.55	13.62	32.28	23.14
30.57	49.84	6.01	7.85	13.86	38.45	28.94
11.13	22.95	5.99	8.07	14.06	40.70	31.60
66.79	225.27	5.43	10.06	15.49	54.19	57.82
21.69	82.90	5.42	10.84	16.26	58.57	67.47
22.27	57.51	5.46	11.15	16.62	63.06	74.17
182.95	221.94	4.95	8.59	13.55	100.00	100.00
0.00	0.00	4.95	8.59	13.55	100.00	ERR
495.30	959.30					

ZINC CIRCUIT

	CUMUL GRADE (Pb)	CUMUL GRADE (Zn)	CUMUL REC. (Pb)	CUMUL REC. (Zn)
Zn CONC 1	4.24	14.30	22.74	38.33
Zn CONC 2	4.46	15.54	30.13	52.44
Zn CONC 3	4.71	15.56	37.71	62.24
TAILS	4.43	8.86	100.00	100.00

APPENDIX 15

Flotation Column Papers

COLUMN FLOTATION PAPERS

PAPER NUMBER	AUTHOR	TITLE	SOURCE
406014	Clingan, B. V. & McGregor, D. R.	Column Flotation Experience at Magma Copper Company	Minerals and Metallurgical Processing, p. 121-125, August 1987
406015	Dobby, G. S. & Finch, J. A.	Flotation Column Scale-up and Modelling	Canadian Institute of Mining and Metallurgy Bulletin, p. 89-96, May 1986
406016	Dobby, G. S. & Finch, J. A.	Particle Collection in Columns - Gas Rate and Bubble Size Effects	Canadian Metallurgy Quarterly, 1986
406017	Dobby, G. S. & Finch, J. A.	Mixing Characteristics of Industrial Flotation Columns	Chemical Engineering Science, vol. 40, No. 7, p. 1061-1068, 1985
406018	Dobby, G. S. & Finch, J. A.	Particle Size Dependence in Flotation Derived from a Fundamental Model of the Capture Process	Unpublished
406019	Hyde, G. R. & Stojsic, A.	Electrochemical and Column Flotation Technology Can Mean Higher Recovery at Lower Operating Costs	Mining Engineering, p. 481-484, July 1987
406020	McKay, J. D., Foot, D. G. & Huiatt, J. L.	Column Flotation of Montana Chromite Ore	Minerals and Metallurgical Processing, p. 170-177, August 1986
406021	Wheeler, D. A.	Column Flotation - The Original Column	Unpublished
406022	Yianatos, J. B., Espinosa-Gomez, R., Finch, J. A., Laplante, A. R. & Dobby, G.S.	Effect of Column Height of Flotation Column Performance	Minerals and Metallurgical Processing, p. 11-14, February 1988
406023	Yianatos, J. B., Finch, J. A. & Laplante, A. R.	Holdup Profile and Bubble Size Distribution of Flotation Column Froths	Canadian Metallurgical Quarterly, p. 1-7, 1986
406024	Yianatos, J. B., Laplante, A. R. & Fince, J. A.	Estimation of Local Holdup in the Bubbling and Froth Zones of a Gas-Liquid Column	Chemical Engineering Science, vol. 40, No. 10, p. 1965-1968, 1985

COLUMN FLOTATION PAPERS

PAPER NUMBER	AUTHOR	TITLE	SOURCE
406025	Yianatos, J. B., Espinosa-Gomez, R. & Finch, J. A.	Column Flotation Manual	Unpublished

APPENDIX 16

Grade/Recovery vs. Particle Size in Column Flotation

PROCESS STUDIES GROUP

technical notes

Project No. H-34214

File No. H-1-E

Project Column Commissioning

Date 01/05/87

Page 1

***Subject*:** Grade/Recovery Versus Particle Size in Column Flotation

***Author(s)*:** B. Blizzard

***Objective*:** Operate column on rougher feed to establish grade/recovery versus particle size relationships.

Discussion:

1. Column grades were higher in every size fraction compared to grades achieved in the plant rougher (see Figure 5). This supports the column's ability to produce a clean concentrate.
2. Overall concentrate grade with column was 67.3% Sb at 67.7% recovery. Plant rougher I achieves a grade of 45.3% Sb at 64.1%. The improved recovery on the column can be related to improved recovery of coarse (+75 micrometer) particles (see Figures 1 to 4).
3. Lower fine recovery in the column indicates a need for additional reagents and/or finer bubble generation.

Methods: Flotation column was commissioned at Durham Mines on plant feed. Samples were taken of feed, concentrate and tails. These samples were sized and assayed.

PARTICLE SIZE ANALYSIS

SAMPLE NAME COLUMN FEED TABLE:H 1-2 DATE : MAY 01

DATA			CALCULATIONS				
PARTICLE SIZE RANGE (micrometer)	SAMPLE WEIGHT (grams)	METAL ASSAY (%)	WEIGHT DISTRIBUTION	WEIGHT PASSING	METAL CONTENT	METAL DISTRIBUTION	METAL PASSING
+149	19.6	0.30	19.5	80.5	5.9	2.95	97.1
100-149	13.4	0.88	13.4	67.1	11.8	5.91	91.1
75-100	10.4	1.82	10.4	56.7	18.9	9.48	81.7
44-75	14.3	2.76	14.3	42.5	39.3	19.78	61.9
37-44	2.9	4.53	2.9	39.6	13.1	6.58	55.3
25-37	4.5	3.64	4.5	35.1	16.3	8.21	47.1
-25	35.2	2.67	35.1	0.0	93.7	47.09	0.0
NA	0.0	0.00	0.0	0.0	0.0	0.00	0.0
NA	0.0	0.00	0.0	0.0	0.0	0.00	0.0
NA	0.0	0.00	0.0		0.0	0.00	
TOTALS	100.3	2.1	100		199.0		
				BALANCE	94.75		

FIGURE 1

PARTICLE SIZE ANALYSIS

SAMPLE NAME COLUMN CONCENTRATE TABLE:H 1-3 DATE : MAY 01

DATA			CALCULATIONS					
PARTICLE SIZE RANGE (micrometer)	SAMPLE WEIGHT (grams)	METAL ASSAY (%)	WEIGHT DISTRIBUTION	WEIGHT PASSING	METAL CONTENT	METAL DISTRIBUTION	METAL PASSING
+149	0.7	40.40	0.5	99.5	18.3	0.27	99.7
100-149	3.8	65.30	2.5	97.1	160.4	2.37	97.4
75-100	12.1	68.40	7.8	89.3	535.0	7.91	89.4
44-75	32.8	69.40	21.2	68.1	1471.4	21.76	67.7
37-44	6.8	69.30	4.4	63.7	304.6	4.51	63.2
25-37	18.1	69.20	11.7	52.0	809.6	11.98	51.2
-25	80.4	66.60	52.0	0.0	3461.3	51.20	0.0
NA	0.0	0.00	0.0	0.0	0.0	0.00	0.0
NA	0.0	0.00	0.0	0.0	0.0	0.00	0.0
NA	0.0	0.00	0.0		0.0	0.00	
TOTALS	154.7	67.3	100		6760.7		
				BALANCE	100.46		

FIGURE 2

PARTICLE SIZE ANALYSIS

SAMPLE NAME COLUMN TAILS TABLE:H 1-4 DATE : MAY 01

DATA			CALCULATIONS				
PARTICLE SIZE RANGE (micrometer)	SAMPLE WEIGHT (grams)	METAL ASSAY (%)	WEIGHT DISTRIBUTION	WEIGHT PASSING	METAL CONTENT	METAL DISTRIBUTION	METAL PASSING
+149	9.6	0.24	11.9	88.1	2.9	3.90	96.1
100-149	9.4	0.41	11.6	76.5	4.8	6.52	89.6
75-100	7.9	0.52	9.8	66.7	5.1	6.95	82.6
44-75	12.6	0.74	15.6	51.1	11.6	15.77	66.9
37-44	1.8	1.26	2.2	48.8	2.8	3.84	63.0
25-37	4.5	0.99	5.6	43.2	5.5	7.54	55.5
-25	34.9	0.94	43.2	0.0	40.7	55.49	-0.0
NA	0.0	0.00	0.0	0.0	0.0	0.00	-0.0
NA	0.0	0.00	0.0	0.0	0.0	0.00	-0.0
NA	0.0	0.00	0.0		0.0	0.00	
TOTALS	80.7	0.69	100		73.3		
				BALANCE	106.17		

FIGURE 3

FIGURE 4

COLUMN FLOTATION ON ROUGHER FEED AT DURHAM

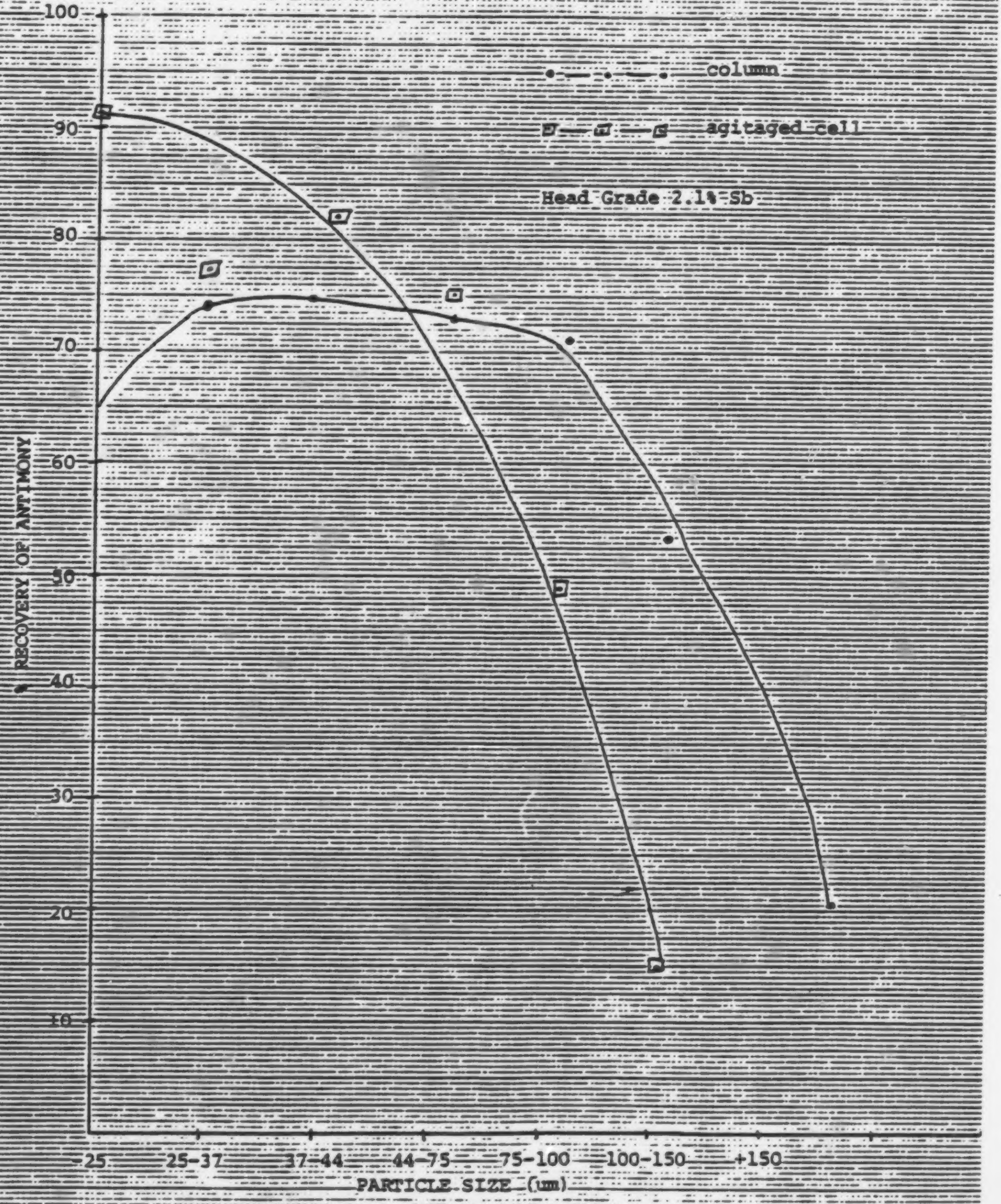
	COLUMN		DURHAM ROUGHER	
SIZE RANGE (um)	GRADE (%)	RECOVERY (%)	GRADE (%)	RECOVERY (%)
+150	40.4	20.1	-	-
100-150	65.3	53.7	19.0	14.4
75-100	68.4	72.0	59.0	50.1
44-75	69.4	73.8	57.6	76.7
37-44	69.3	73.5	56.2	82.5
25-37	69.2	73.8	48.0	78.9
-25	66.6	65.6	40.9	91.9
OVERALL	67.3	67.7	45.3	64.1

COLUMN OPERATING PARAMETERS:

FEED RATE: .07 tph
AIR PRESSURE 9 psig
REAGENTS: PLANT ADDITION

FIGURE 5

% RECOVERY ON ROUGHER FEED



APPENDIX 17

Operation and Control of Column Float

PROCESS STUDIES GROUP

technical notes

Project No. H-34214
File No. H-1
Project Column Commissioning
Date 25/09/87
Page 1

Subject: Operation and Control of Column Float

Author(s): I. Noddin

Objective: Set up and operate column flotation to see what affect operating parameters have on grade and recoveries.

Discussion: The need to be able to easily and accurately control all operating variables seemed very important because the slightest change in any parameter affected the column. When any variable was changed, the affect on the column was almost instant.

The column could be easily changed to achieve better grades or recoveries The low feed grade had a great effect on the column, the operation had to be switched to a rougher operation pulling harder to achieve recovery where grade was lost.

Further testing will have to be done to see the effect of all operating parameters.

Methods: Column flotation was operated on rougher feed at Durham Resources. Operating variables were adjusted and samples taken on feed concentration and tail. Results are presented in Figure 1.

FIGURE 1

TEST	TAIL RATE	FEED RATE	FEED % Sb	TAIL % Sb	CONC % Sb	RINSE	RECOVERY	AIR PRESSURE	
1	3.5% .39m^3/h .014t/h	5.0% .36m^3/h .02t/h	4.43	1.71	65.6	.03m^3/h	63.04%	10 psi	Column was operated for grade showing loss of recovery.
2	16.0% .32m^3/h .057t/h	17.5% .3m^3/h .059t/h	2.92	1.92	64.92	.019m^3/h	35.28%	10 psi	To maintain a grade at lower feeds, recovery loss increased.
3	24.5% .11m^3/h .03199t/h	33.0% .076m^3/h .03182t/h	2.16	0.54	56.96	.036m^3/h	75.6%	17.5 psi	When column was set to rougher operation, recovery increased but conc. decreased.

APPENDIX 18

Operation and Control of Column Flotation

PROCESS STUDIES GROUP	technical notes	Project No. H-34214	File No. H-1
	Project Column Commissioning	Date 13/10/87	Page 1

Subject: Operation and Control of Column Flotation

Author(s): I. Noddin

Objective: Set up and operate column flotation to see what affect operating parameters have on grade and recoveries.

Discussion:

1. Increasing the air pressure shows an increase in recovery, but 16 psi seemed to be the best pressure because at lower pressures it was difficult to keep a constant froth bed level and at higher pressures, the froth was forced off instead of producing compacted froth (see Figure 5).
2. Longer residence time showed a decrease in recovery, this is because the flow and tonnage was too low, not allowing enough ore to have a good controllable froth (see Figures 1 and 2).
3. Setting parameters for a greater recovery showed a decrease in the grade of the concentrate (see Figures 2 and 4).
4. When the density of the column was decreased, the recovery also decreased because the amount of ore in the column was too low to have a good steady controllable froth bed (see Figure 6).
5. When the grade of the feed decreased, it was difficult to operate because there was not enough floatable ore to keep a froth bed.

Methods: The column was operated on rougher feed at Durham Resources, operating variables were adjusted and samples taken of feed, concentrate, and tail. Data obtained is presented in Figures 1 and 2.

FIGURE 1
COLUMN COMMISSIONING OPERATING PARAMETERS

TEST	DATE	% SOLIDS	FLOW m^3/h	t/h	AIR FLOW PSI	RINSE FLOW	RESIDENCE TIME	% Sb FEED	% Sb CONC	% Sb TAIL	% RECOVERY	
1	OCT 1	34	.173	.075	19	–	3.3	2.55	64.6	0.87	66.6	
2	"	34	.173	.075	13	–	3.3	"	68.0	1.25	51.9	
3	"	35	.162	.073	16	–	3.6	"	68.5	1.09	58.1	
4	"	19	.162	.035	16	–	3.6	"	65.5	1.54	40.6	
1	OCT 6	43	.068	.04	16	–	8.5	2.55	67.7	1.60	38.5	
2	"	32	.28	.112	16	–	2.1	"	68.5	1.10	57.8	
3	"	20	.162	.037	16	–	3.6	"	67.3	1.44	44.5	
4	"	12	.162	.021	16	–	3.6	"	67.4	1.83	29.0	
1	OCT 7	33	.173	.075	16	–	3.3	3.38	60.0	2.29	33.4	
2	"	34	.173	.075	9	–	3.3	"	69.5	1.62	53.3	
3	"	42	.036	.045	16	–	16.0	"	68.8	2.56	25.1	
4	"	15	.173	.075	16	–	3.3	"	67.9	2.31	32.8	
5	"	34	.09	.027	19	–	6.4	3.37	66.71	1.009	71.1	ran for grade
6	"	25	.18	.078	16	–	7.4	"	67.58	0.97	72.3	ran for recovery

FIGURE 2

ASSAYS % Sb

DATE	NO.	XMET	KT	TEST NO.	SAMPLE
Oct 1	C-1	63.22	64.60	1	CONC
"	C-2	65.68	68.00	2	CONC
"	C-3	65.17	68.50	3	CONC
"	C-4	65.6	65.50	4	CONC
"	C-5	0.85	0.87	1	TAIL
"	C-6	1.1	1.25	2	TAIL
"	C-7	1.06	1.09	3	TAIL
"	C-8	1.40	1.54	4	TAIL
"	C-9	2.41	2.55		FEED
Oct 6	C-10	1.21	1.60	1	TAIL
"	C-11	0.97	1.10	2	TAIL
"	C-12	1.32	1.44	3	TAIL
"	C-13	1.63	1.83	4	TAIL
"	C-14	66.26	67.70	1	CONC
"	C-15	69.78	68.50	2	CONC
"	C-16	68.03	67.30	3	CONC
"	C-17	67.22	67.40	4	CONC
"	C-18	2.55	2.55		FEED
Oct 7	C-19	1.64	2.29	1	TAIL
"	C-20	1.39	1.62	2	TAIL
"	C-21	1.89	2.56	3	TAIL
"	C-22	1.72	2.31	4	TAIL
"	C-23	67.82	68.00	1	CONC
"	C-24	65.43	69.50	2	CONC
"	C-25	64.96	68.80	3	CONC
"	C-26	65.14	67.90	4	CONC
"	C-27	3.54	3.38		FEED

FIGURE 3

RECOVERY/RESIDENCE TIME

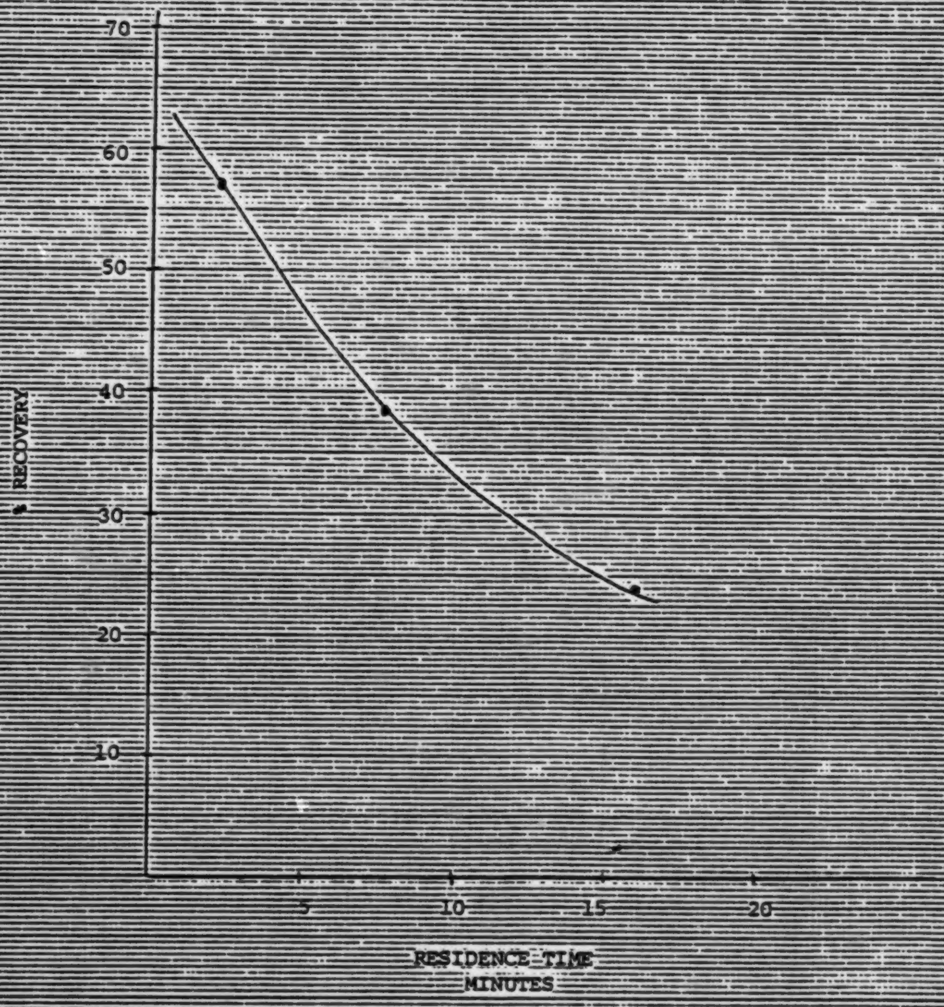


FIGURE 4

COLUMN FLOTATION
RECOVERY/GRADE

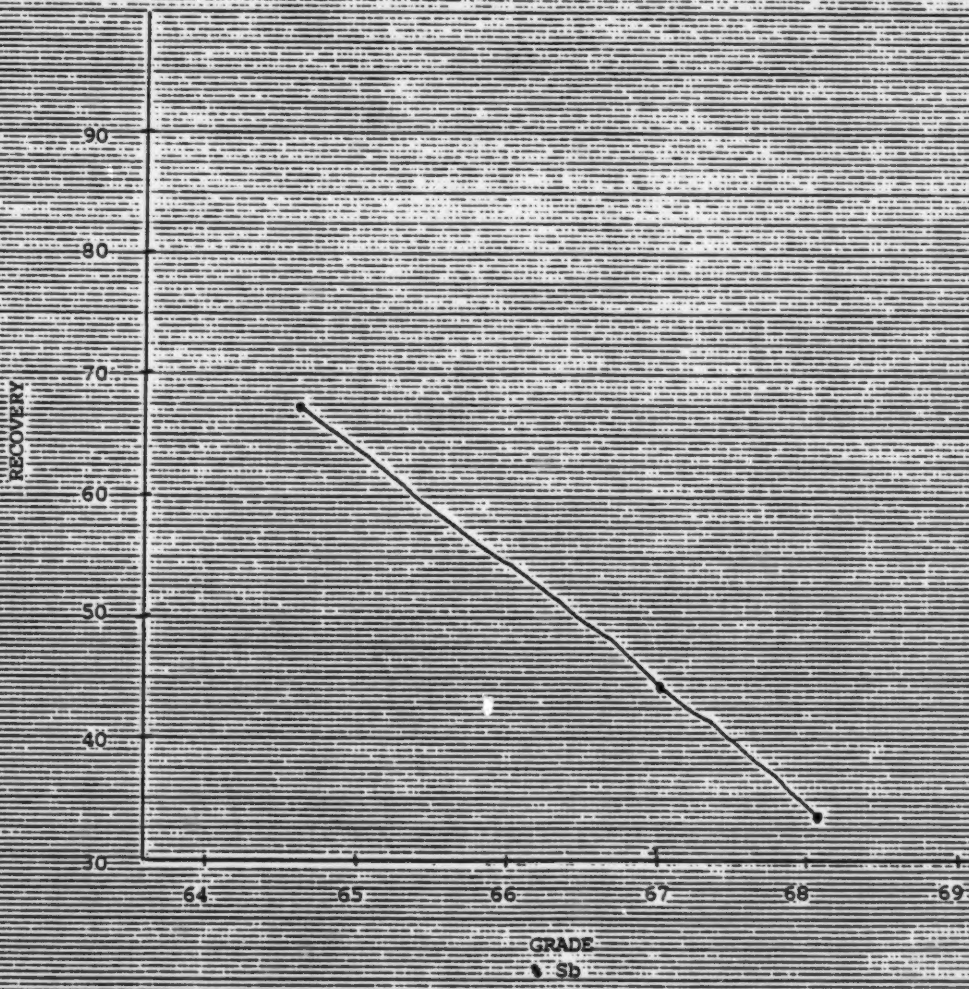


FIGURE 5

AIR PRESSURE/RECOVERY

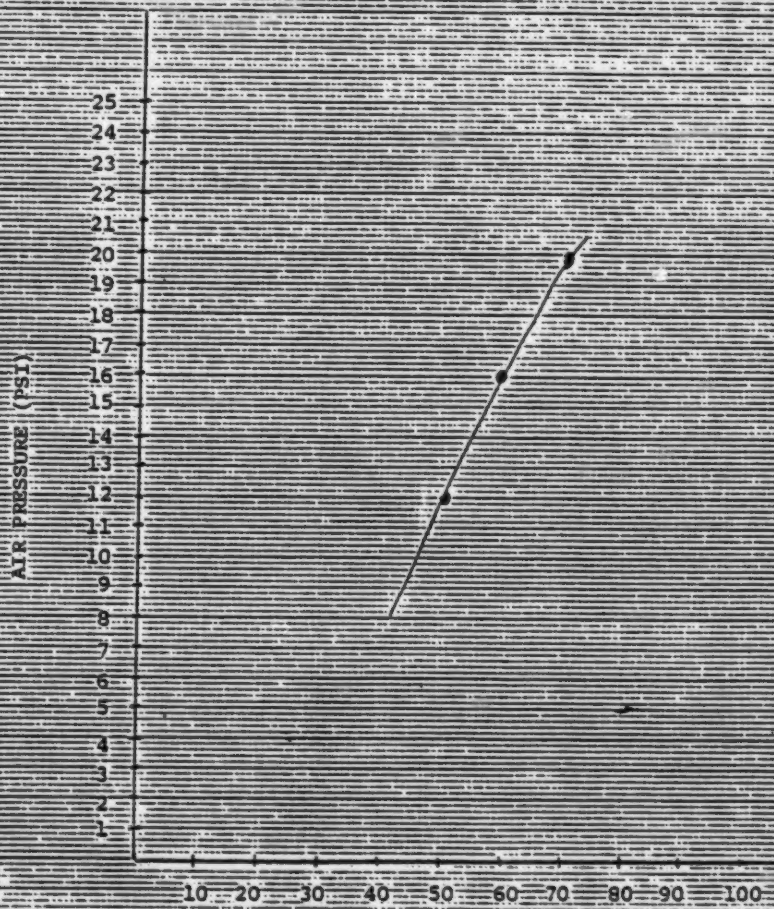
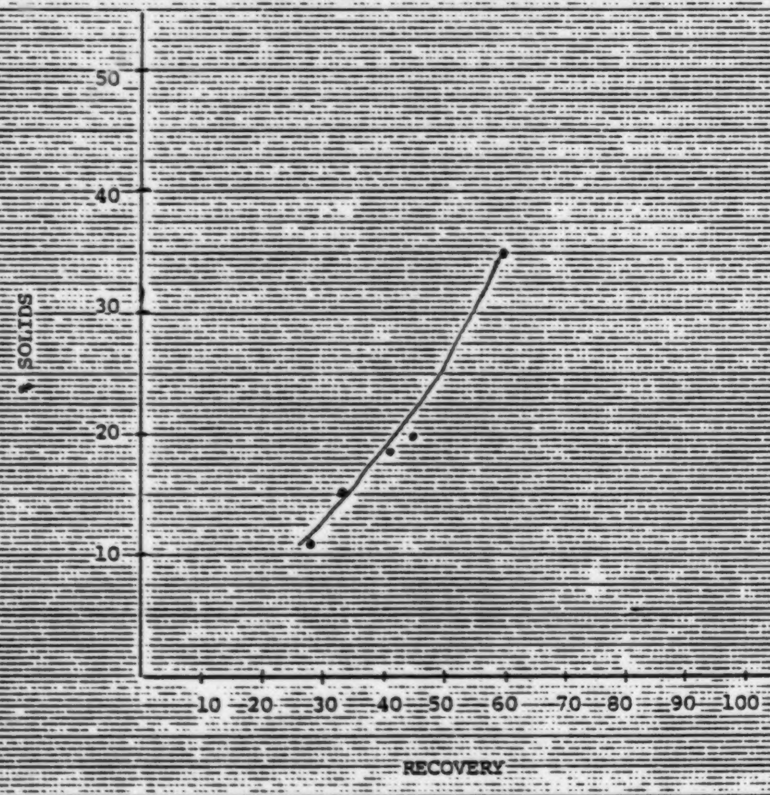


FIGURE 6

COLUMN FLOTATION
% SOLIDS/RECOVERY



APPENDIX 19

Operating Variables

PROCESS STUDIES GROUP	**technical notes**	Project No. H-34214	File No. H-2A
	Project Column Commissioning	Date 14/10/87	Page 1

Subject: Operating Variables

Author(s): I. Noddin

Objective: To draw up graphs to have some operating guidelines.

Discussion: Figure 1 shows the flowrates required to achieve various residence times in a 3.20 m and 0.062 m column float cell.

Figures 2 and 3 show flowrates and tonnages achieved using a variable speed pump. Because of the need to lower densities, additional water could be added along the feed line using a flowmeter to control water (Figure 4).

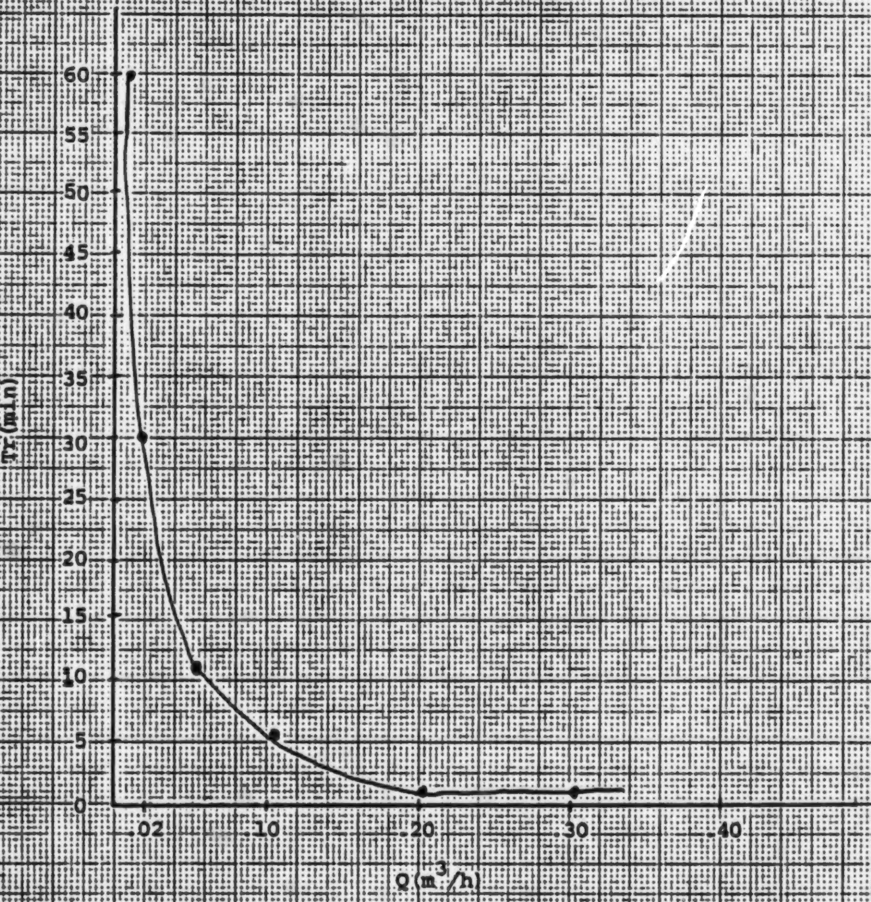
To achieve a constant flowrate withOUT line sanding, Figure 5 shows size of line required to pump at 18, 12 and 6 foot head levels.

Air pressure to the column can be monitored by using a pressure gauge from 0 to 30 psi.

Methods: Flowrates were measured and graphs were drawn up to have calibration curves for various pump and flowmeters. Calculations were done to achieve proper feed lines and column size.

FIGURE 1

COLUMN RESIDENCE TIME

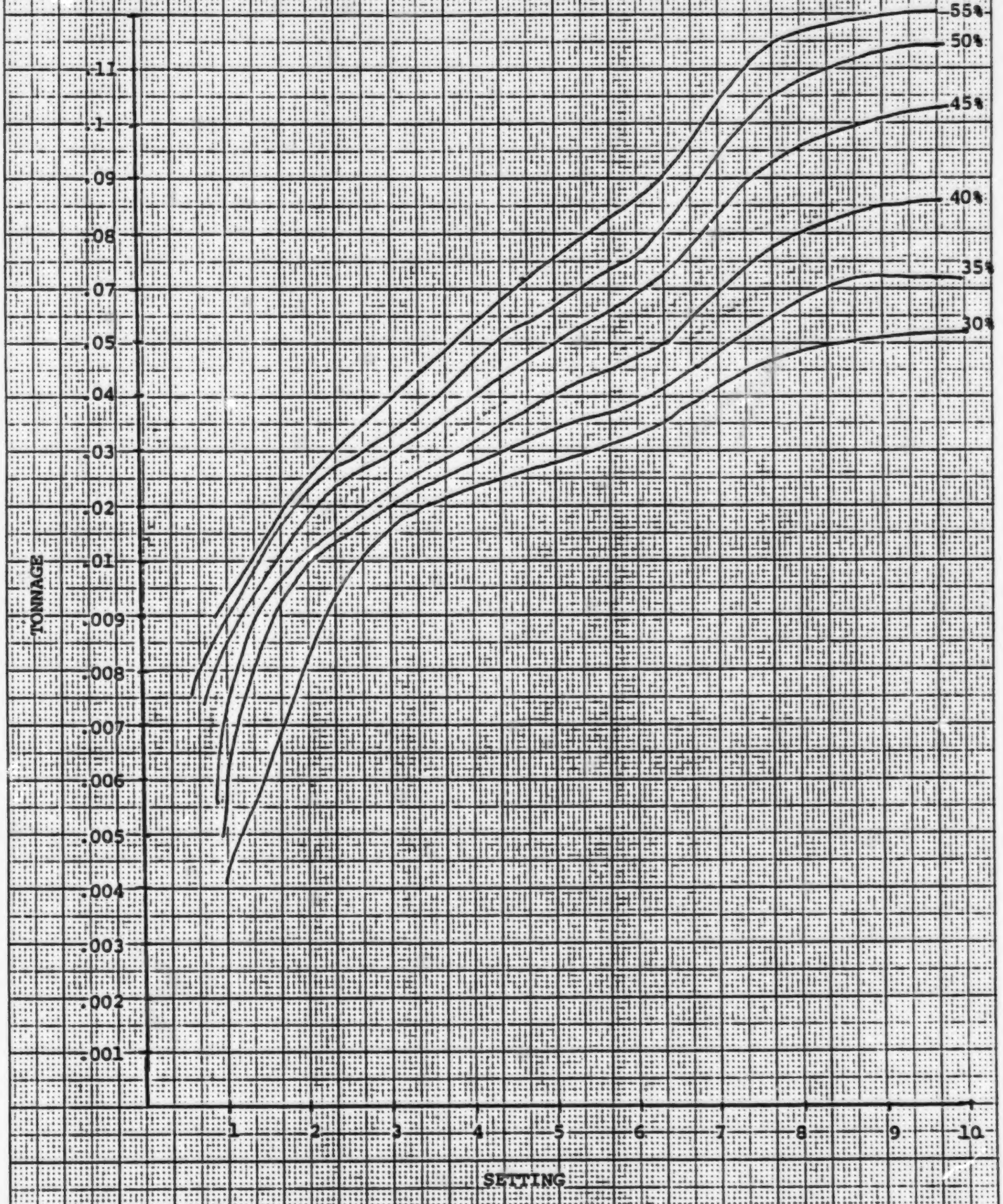


$$\text{Volume} = \text{Length } 3.20^{m} \left(\frac{\pi\ \text{diameter}^{2}\ .062^{m}}{4}\right) = 0.0096\ m^{3}$$

$$Tr = \frac{\text{Volume } 0.0096\ m^{3}}{Q(m^{3}/h)}$$

METRIC

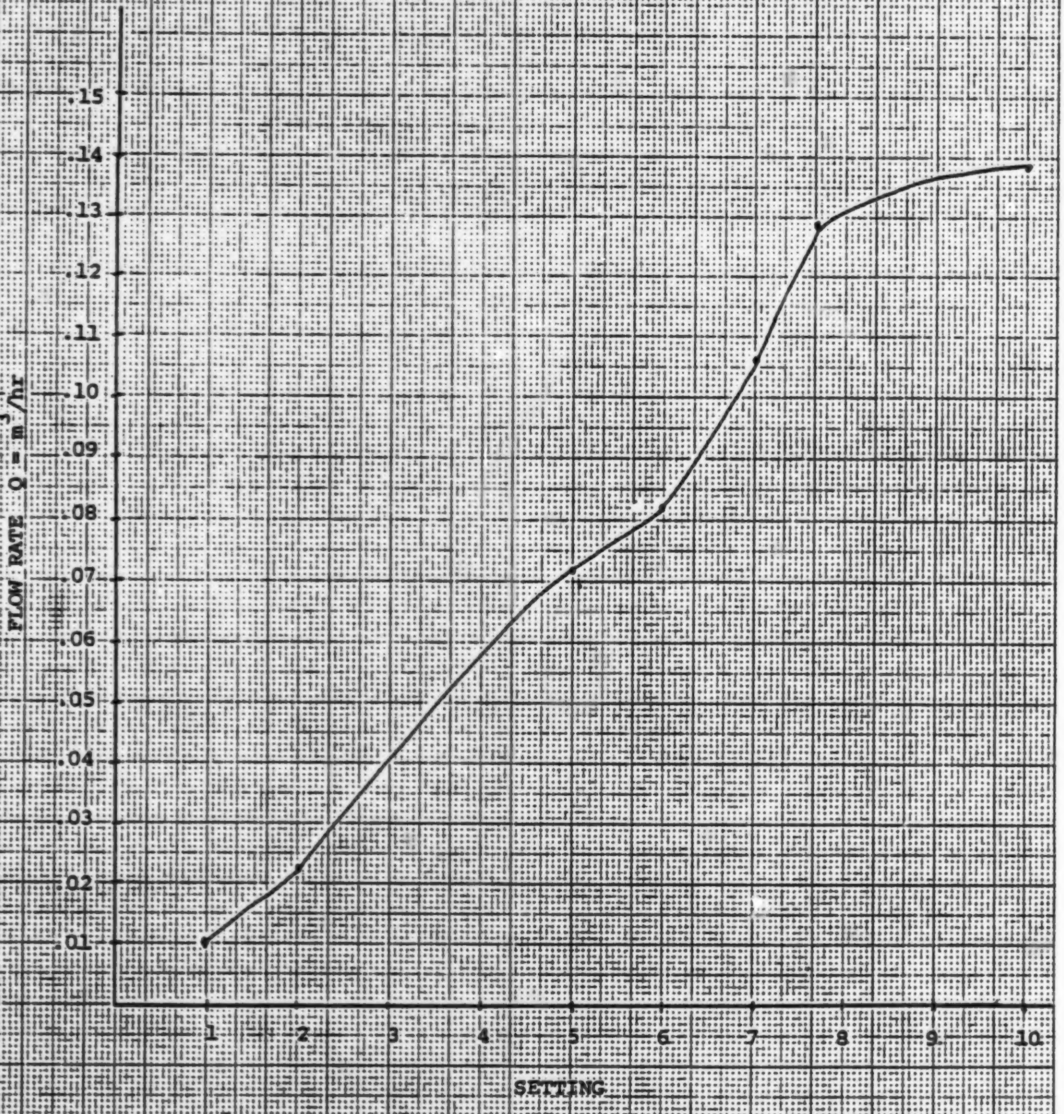
FIGURE 2



GRAPHIC CONTROLS CANADA L
GRFC14 SQUARE 10 X 10 TO THE CM
SPECIFY TRACING OR DRAWING PAPER

FIGURE 3

6-600 R.P.M. PUMP CALIBRACTION GRAPH
18-HEAD-WATER



METRIC

FIGURE 4

FLOW METER CALIBRATION GRAPH - WATER

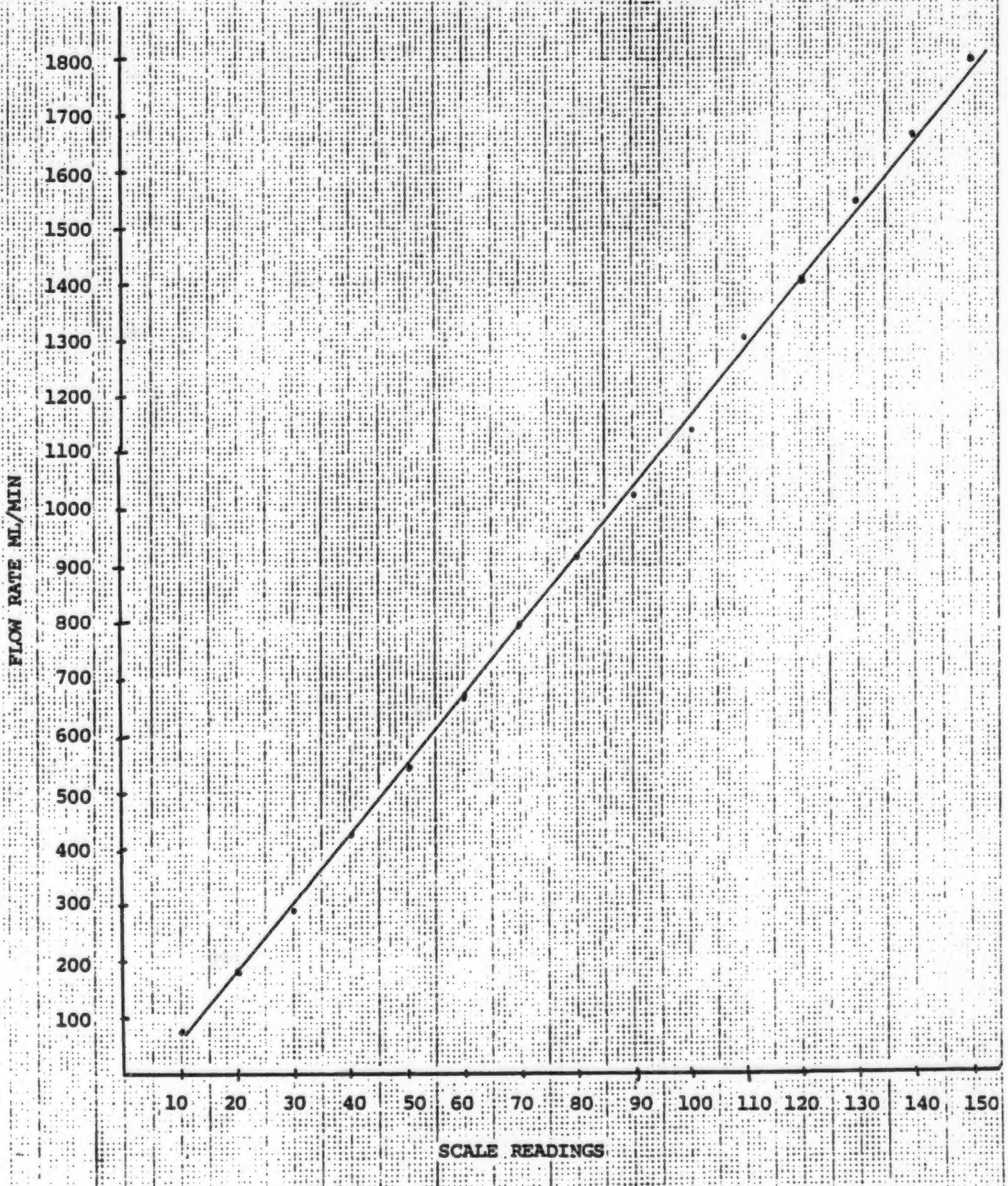
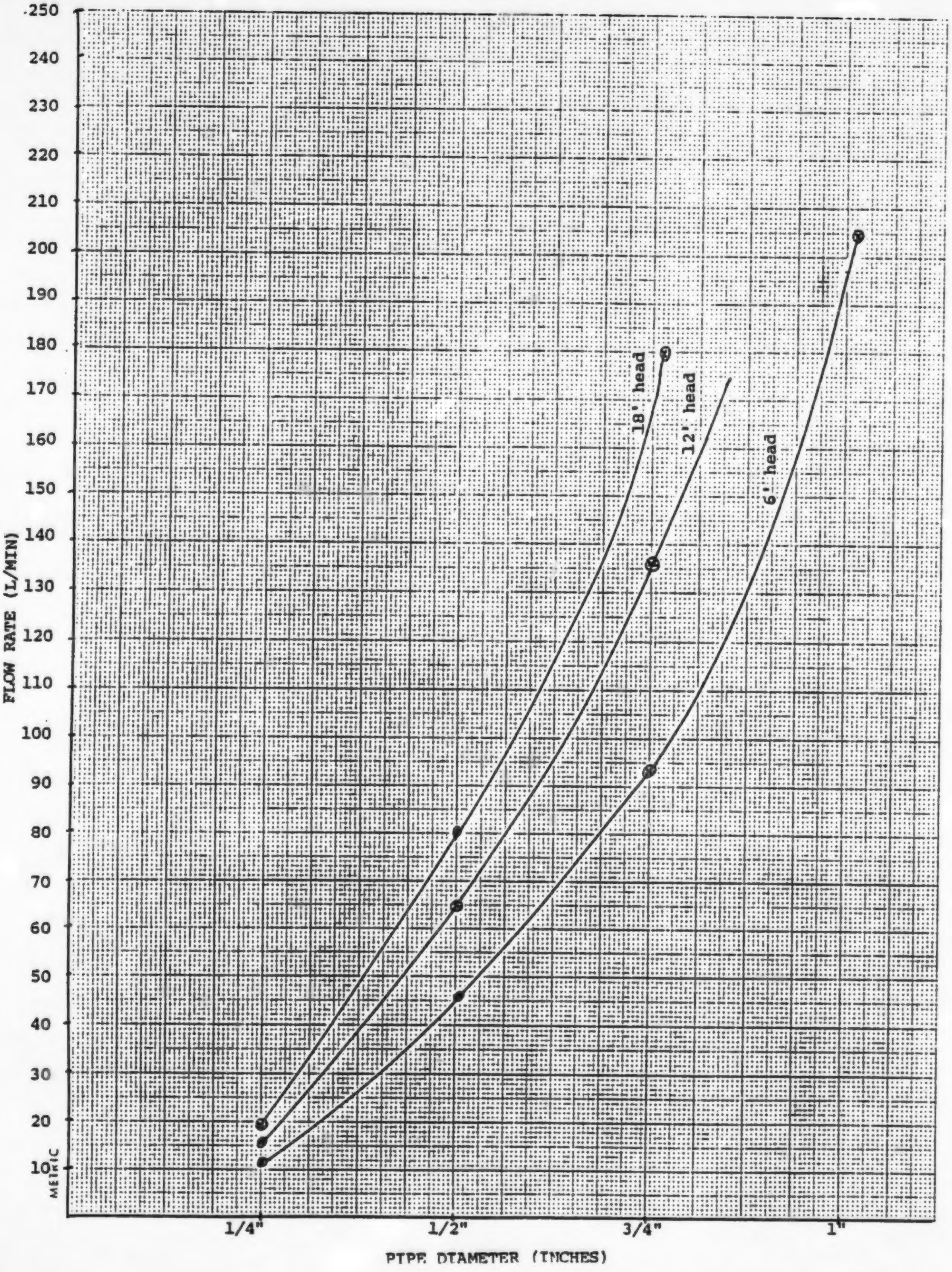


FIGURE 5


.250
240
230
220
210
200
190
180
170
160
150
140
130
120
110
100
90
80
70
60
50
40
30
20
10
METRIC
FLOW RATE (L/MIN)
18' head
12' head
6' head
1/4"
1/2"
3/4"
1"
PIPE DIAMETER (INCHES)

APPENDIX 20

Bubble Generation

PROCESS STUDIES GROUP	*technical notes*	*Project No.* E-76815	*File No.* E-19
	Project Column Flotation	*Date* 02/11/87	*Page* 1

***Subject*:** Bubble Generation

***Author(s)*:** I. Noddin

***Objective*:** To see if the size of the bubbles can be reduced using the sparger now in use.

Discussion: Without the addition of the frother, the size of the bubbles appeared to be about 8 to 10 mm in size.

When the frother (MIBC) was added, it kept the bubbles formed by the sparger from collecting and making the larger bubbles. The size of the smaller bubbles was about 2 to 3 μm in size and seemed to increase in number. The size stayed constant throughout the entire length of the column.

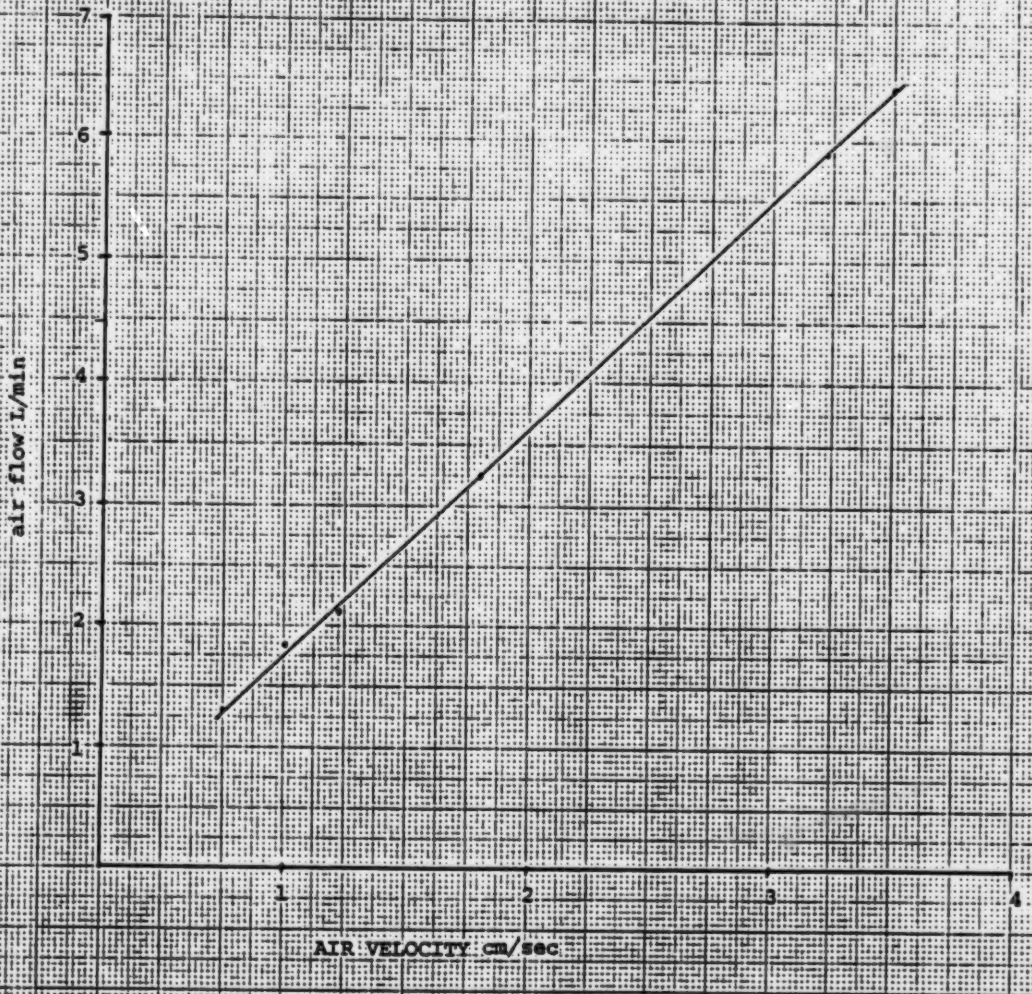
The sparger should be designed to allow frother to be added to the air and further testing should be done to see if this will help recovery of the fines.

The volume of the air may also be important in the recovery. Figure 1 shows the velocity of the air at various air flows within the column.

Methods: The column was filled with water to the feed pOrt, the air pressure was set at 10 psi and 3 mL of frother was added through the tails line.

FIGURE 1

AIR FLOW IN COLUMN



METRIC

APPENDIX 21

Column Recovery by Fraction

PROCESS STUDIES GROUP

technical notes		*Project No.* E-76815	*File No.* E-19
Project Column Flotation		*Date* 02/11/87	*Page* 1

Subject: Column Recovery by Fraction

Author(s): I. Noddin

Objective: To see the effect of grade and recovery by size fraction on rougher feed at Durham Resources.

Discussion: Test 6, which was run for recovery, showed an expected drop in grade and an increase in recovery of the -25 µm fraction. Test 5 has a better concentration grade than test 6; however, it shows a drop in recovery of the fine fractions. Test 6 also has a significantly higher amount of -100 µm Sb in the concentration than test 5, which is shown in the metal distribution (See Figures 1 to 5).

The recovery of both tests, in the -37 µm size, was very poor. Testing should be done to see if the bubble size will affect the recovery of the fine material (See Figures 6 to 8).

The recovery of the -74 µm was also poor. This is due to possibly to poor liberation of the ore in the coarse fractions.

Methods: Two tests were run; one for grade and one for recovery on rougher feed. Samples were taken of feed, concentration and tail. These samples were screened, cyclosized and sent for assay.

PARTICLE SIZE ANALYSIS

SAMPLE NAME CONC TEST 5 GRADE

DATE : OCT 16/87

DATA

CALCULATIONS

PARTICLE SIZE RANGE (micrometer)	SAMPLE WEIGHT (grams)	METAL ASSAY (%)	WEIGHT DISTRIBUTION	WEIGHT PASSING	METAL CONTENT	METAL DISTRIBUTION	METAL PASSING	REC
+1000	0.00	0.00	0.00	100.00	0.0	0.00	100.0	
600-1000	0.00	0.00	0.00	100.00	0.0	0.00	100.0	
300-600	0.0	0.00	0.00	100.00	0.0	0.00	100.0	
150-300	0.0	0.00	0.00	100.00	0.0	0.00	100.0	
144.00	0.6	61.30	0.73	99.27	44.7	0.64	99.4	41.4
105.00	3.30	65.60	4.01	95.26	263.4	3.77	95.6	70.8
74.00	6.9	68.20	8.39	86.86	572.5	8.20	87.4	84.7
53.00	9.2	69.60	11.19	75.67	779.0	11.16	76.2	86.1
37.00	9.9	69.00	12.04	63.63	831.0	11.91	64.3	
-37.00	52.3	68.80	63.63					
TOTALS	82.20	69.80	100.00		6980.0			
CYCLOSIZER								
38	4.90	70.20	14.85	48.78	1042.18	14.93	49.39	85.8
28	1.30	69.30	3.94	44.84	272.95	3.91	45.48	73.9
20.4	2.50	68.60	7.57	37.27	519.61	7.44	38.03	65.8
13.8	2.70	68.90	8.18	29.09	563.63	8.07	29.96	64.7
10.8	1.40	68.10	4.24	24.84	288.86	4.14	25.82	68.6
-10.8	8.20	72.54	24.84	0.00	1802.19	25.82	0.00	
WT. CYC.	21.00				6980.00			
HEAD		69.80						

FIGURE 1

FIGURE 2

PARTICLE SIZE ANALYSIS

SAMPLE NAME COLUMN FEED

DATE : OCT 16/87

DATA			CALCULATIONS				
PARTICLE SIZE RANGE (micrometer)	SAMPLE WEIGHT (grams)	METAL ASSAY (%)	WEIGHT DISTRIBUTION	WEIGHT PASSING	METAL CONTENT	METAL DISTRIBUTION	METAL PASSING
+1000	0.00	0.00	0.00	100.00	0.0	0.00	100.0
600-1000	0.00	0.00	0.00	100.00	0.0	0.00	100.0
300-600	0.0	0.00	0.00	100.00	0.0	0.00	100.0
150-300	0.0	0.00	0.00	100.00	0.0	0.00	100.0
144.00	20.1	0.51	15.73	84.27	8.0	2.45	97.5
105.00	13.90	1.05	10.88	73.40	11.4	3.49	94.1
74.00	11.8	2.53	9.23	64.16	23.4	7.14	86.9
53.00	9.9	3.47	7.75	56.42	26.9	8.22	78.7
37.00	10.8	5.83	8.45	47.97	49.3	15.07	63.6
-37.00	61.3	4.59	47.97				
	-------		-------		-------		
TOTALS	127.80	3.27	100.00		327.0		
CYCLOSIZER							
38	0.90	17.50	2.11	45.86	36.85	11.27	52.35
28	1.80	3.53	4.21	41.65	14.87	4.55	47.81
20.4	2.50	3.20	5.85	35.80	18.72	5.72	42.08
13.8	3.00	3.40	7.02	28.78	23.87	7.30	34.79
10.8	1.50	3.44	3.51	25.27	12.07	3.69	31.09
-10.8	10.80	4.02	25.27	0.00	101.67	31.09	0.00
WT. CYC.	20.50				327.00		
HEAD		3.27					

PARTICLE SIZE ANALYSIS

SAMPLE NAME TAIL TEST 5 DATE : OCT 16/87

DATA			CALCULATIONS				
PARTICLE SIZE RANGE (micrometer)	SAMPLE WEIGHT (grams)	METAL ASSAY (%)	WEIGHT DISTRIBUTION	WEIGHT PASSING	METAL CONTENT	METAL DISTRIBUTION	METAL PASSING
+1000	0.00	0.00	0.00	100.00	0.0	0.00	100.0
600-1000	0.00	0.00	0.00	100.00	0.0	0.00	100.0
300-600	0.0	0.00	0.00	100.00	0.0	0.00	100.0
150-300	0.0	0.00	0.00	100.00	0.0	0.00	100.0
144.00	19.4	0.30	15.77	84.23	4.7	4.15	95.8
105.00	14.10	0.31	11.46	72.76	3.6	3.12	92.7
74.00	11.4	0.40	9.27	63.50	3.7	3.25	89.5
53.00	8.5	0.63	6.91	56.59	4.4	3.82	85.7
37.00	10.4	0.74	8.46	48.13	6.3	5.49	80.2
-37.00	59.2	1.61	48.13				
	-------		-------		-------		
TOTALS	123.00	1.14	100.00		114.0		
CYCLOSIZER							
38	0.80	3.91	1.88	46.25	7.34	6.44	73.73
28	1.80	1.06	4.23	42.03	4.48	3.93	69.80
20.4	2.70	1.12	6.34	35.69	7.10	6.23	63.57
13.8	2.70	1.25	6.34	29.35	7.92	6.95	56.62
10.8	1.40	1.20	3.29	26.06	3.94	3.46	53.16
-10.8	11.10	2.33	26.06	0.00	60.61	53.16	-0.00
WT. CYC.	20.50				114.00		
HEAD		1.14					

FIGURE 3

PARTICLE SIZE ANALYSIS

SAMPLE NAME TAIL TEST 6 RECOVERY

DATE : OCT 16/87

DATA			CALCULATIONS					
PARTICLE SIZE RANGE (micrometer)	SAMPLE WEIGHT (grams)	METAL ASSAY (%)	WEIGHT DISTRIBUTION	WEIGHT PASSING	METAL CONTENT	METAL DISTRIBUTION	METAL PASSING	
+1000	0.00	0.00	0.00	100.00	0.0	0.00	100.0	
600-1000	0.00	0.00	0.00	100.00	0.0	0.00	100.0	
300-600	0.0	0.00	0.00	100.00	0.0	0.00	100.0	
150-300	0.0	0.00	0.00	100.00	0.0	0.00	100.0	
144.00	15.4	0.38	15.10	84.90	5.7	4.59	95.4	25.6
105.00	11.10	0.57	10.88	74.01	6.2	4.96	90.4	46.1
74.00	9.9	0.95	9.71	64.31	9.2	7.38	83.1	63.4
53.00	6.5	1.37	6.37	57.93	8.7	6.99	76.1	61.8
37.00	8.5	1.77	8.33	49.60	14.8	11.80	64.3	
-37.00	50.6	1.71	49.60					
	-------		-------		-------			
TOTALS	101.98	1.25	100.00		125.0			
CYCLOSIZER								
36.60	0.50	5.11	1.23	48.37	6.27	5.02	59.26	77.9
26.90	2.00	1.03	4.91	43.46	5.06	4.05	55.21	71.9
19.60	2.50	0.89	6.14	37.32	5.46	4.37	50.84	73.2
13.30	2.90	1.02	7.12	30.20	7.26	5.81	45.03	65.2
10.40	1.40	1.12	3.44	26.76	3.85	3.08	41.95	68.7
-10.40	10.90	1.96	26.76	0.00	52.44	41.95	0.00	
WT. CYC.	20.20				125.00			
HEAD		1.25						

FIGURE 4

FIGURE 5

PARTICLE SIZE ANALYSIS

SAMPLE NAME CONC TEST 6

DATE : OCT 16/87

DATA			CALCULATIONS				
PARTICLE SIZE RANGE (micrometer)	SAMPLE WEIGHT (grams)	METAL ASSAY (%)	WEIGHT DISTRIBUTION	WEIGHT PASSING	METAL CONTENT	METAL DISTRIBUTION	METAL PASSING
+1000	0.00	0.00	0.00	100.00	0.0	0.00	100.0
600-1000	0.00	0.00	0.00	100.00	0.0	0.00	100.0
300-600	0.0	0.00	0.00	100.00	0.0	0.00	100.0
150-300	0.0	0.00	0.00	100.00	0.0	0.00	100.0
144.00	0.3	60.70	0.86	99.14	52.3	0.80	99.2
105.00	0.40	60.70	1.15	97.99	69.8	1.07	98.1
74.00	2.4	66.30	6.90	91.09	457.2	7.00	91.1
53.00	2.5	65.70	7.18	83.91	472.0	7.23	83.9
37.00	4.6	66.00	13.22	70.69	872.4	13.36	70.5
-37.00	24.6	63.10	70.69				
	-------		-------		-------		
TOTALS	34.80	65.30	100.00		6530.0		
CYCLOSIZER							
36.60	3.40	68.30	13.13	57.56	897.02	13.74	56.80
26.90	2.10	64.60	8.11	49.44	524.03	8.02	48.78
19.60	3.70	63.10	14.29	35.15	901.85	13.81	34.97
13.30	2.70	62.20	10.43	24.72	648.72	9.93	25.03
10.40	1.20	61.80	4.64	20.09	286.47	4.39	20.65
-10.40	5.20	67.12	20.09	0.00	1348.17	20.65	0.00
WT. CYC.	18.30				6530.00		
HEAD		65.30					

FIGURE 6

TEST 5 FOR GRADE

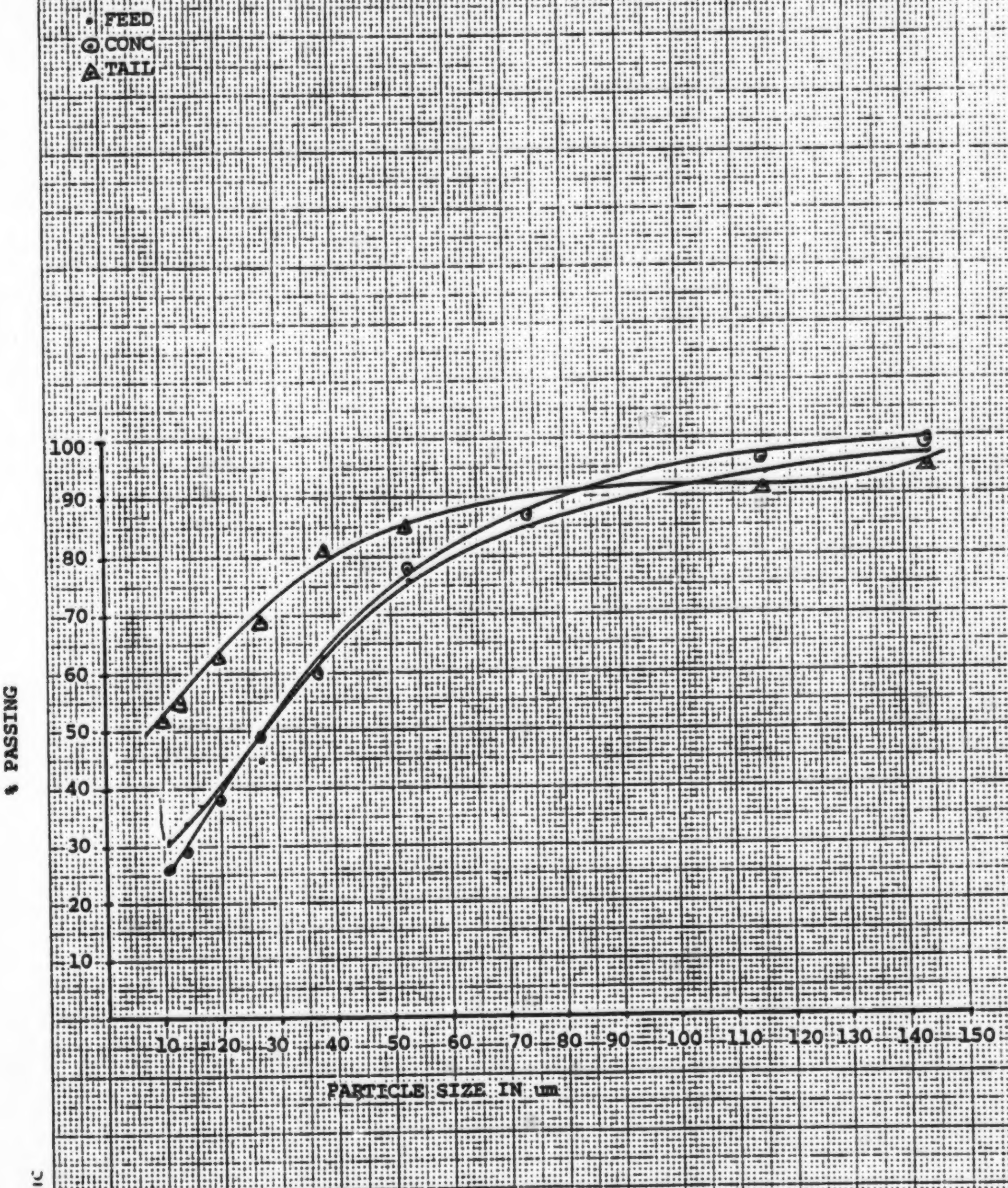
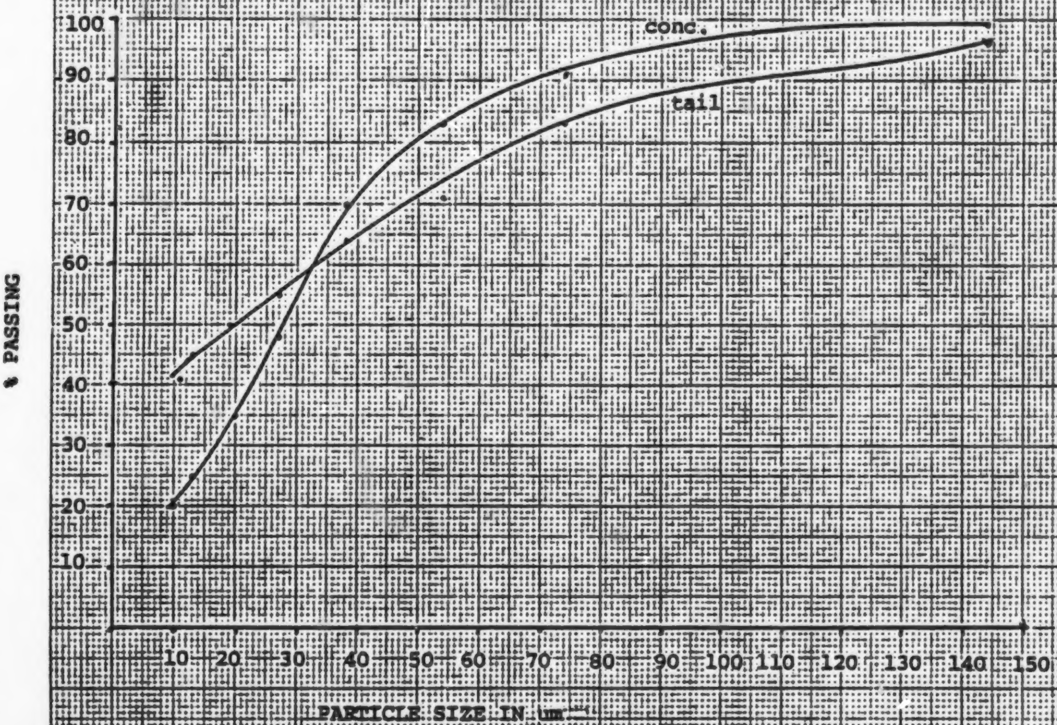


FIGURE 7

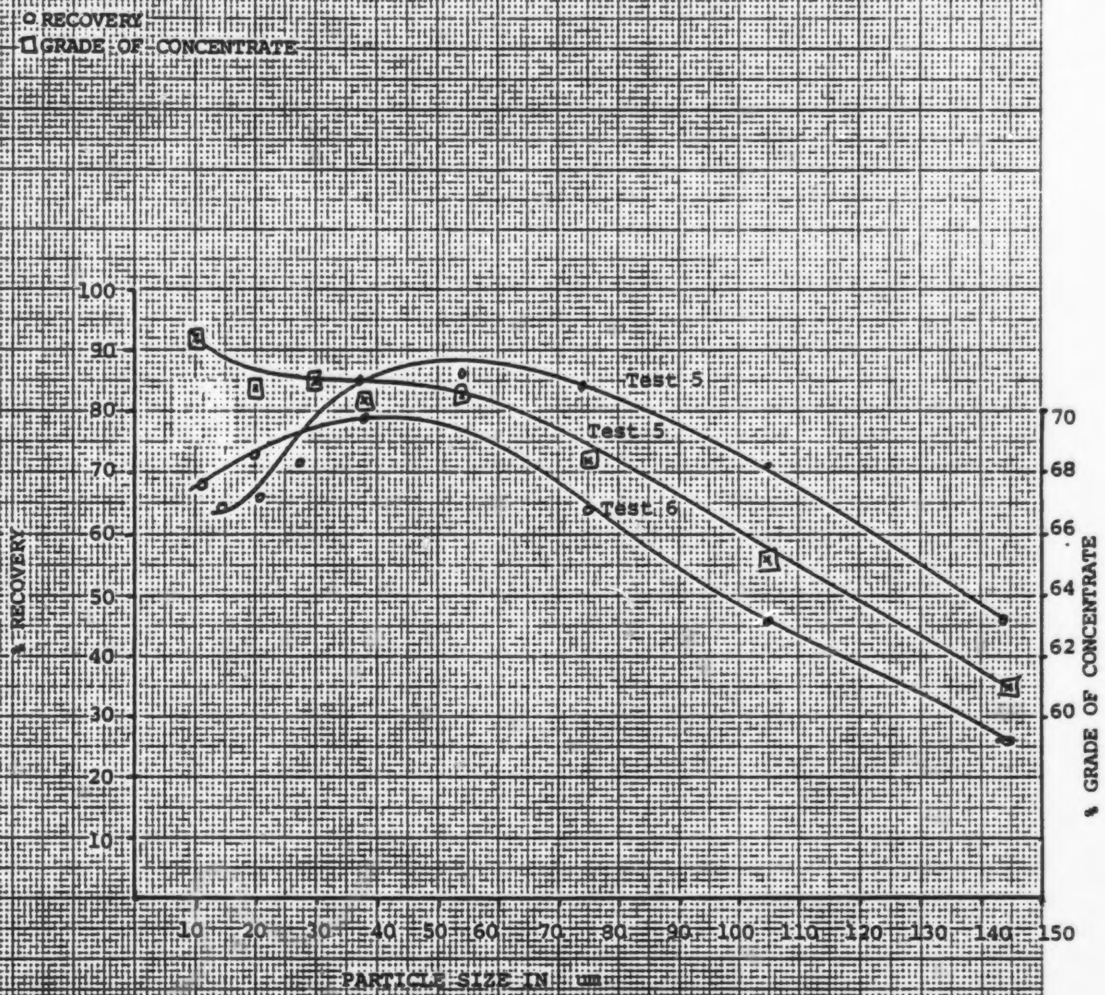
METAL PASSING
TEST 6 FOR RECOVERY



METRIC

FIGURE 8

RECOVERY BY PARTICLE SIZE



METRIC

APPENDIX 22

Column Flotation

PROCESS STUDIES GROUP	*technical notes*	*Project No.*	*File No.*
	Project	*Date* 30/08/88	*Page* 1

Subject: Column Flotation

Author(s): Caroline Arseneau

Objective:

1. To find a procedure to scale-up a laboratory flotation column to a plant size production column.
2. To test the flotation column at Dominion Explorers Inc. mine using various equipment modifications to produce better grades and higher recoveries.

Conclusions:

1. A scale-up procedure provided by Dobby and Finch has been used to develop the mathematics to scale a laboratory column to plant production columns. To facilitate the scale-up procedure, a computerized simulator has been developed by Dobby and his associates.
2. We were unable to produce acceptable running conditions for the column, and therefore were unable to prove that the modified column worked better. More test work is needed in this problem area.

Author(s) C. Arseneau	***technical notes***	*Project No.*	*Page* 2	*of* 6

Discussion: 1.1 INTRODUCTION

It is important that we identify and market all of New Brunswick's resources. By optimizing the grades and recoveries of our province's ore, we make them more marketable.

Often in the mining industry, tailings may contain a lot of high-grade, but fine, material which is too small to be selected during the flotation process. For this reason, a flotation column (Figure 1) is tested to see if it can pick up the fine particles because of its selectiveness in recovering them. Flotation columns are counter-current columns that are presently being applied to froth flotation separation of minerals.

Three main physical differences between the flotation column and conventional flotation, which play a role in producing higher grade concentrates, are the following:

- The cleaning of the froth layer by addition of wash water.
- The increase in froth height.
- Continuous counter-current flow of pulp and bubbles.

Other advantages of the column are as follows:

- Decreases floor space required.
- Energy efficiency increases.
- Lack of turbulence, which gives rise to more controlled, still water conditions for separation.
- Easily adaptable to computer control because column recovery is based on the flows into and out of the column.

Conventional flotation machines would require several sequential cleaning stages, whereas a flotation column can upgrade a fine sized concentrate in a single step.

Author(s) C. Arseneau	technical notes	Project No.	Page 3 of 6

Discussion: 1.2 GRADE AND RECOVERY IMPROVEMENTS

The flotation column basically consists of two distinct zones; the cleaning zone (froth zone) and the collection zone. As shown in Figure 1, for a laboratory or pilot column, the two zones are divided by the point where the feed enters, but the interface lines make the distinction between the two zones for a plant size column.

A bubble swarm, which is produced by a gas sparger located near the bottom of the column, collides counter-currently with the feed particles and selectively entrains the hydrophobic material to the cleaning zone. The hydrophilic and less hydrophobic particles are then removed from the bottom of the column in the tailings stream. We used a bubble generator built from the U.S. Bureau of Mines design, but we were never able to make it work properly because we lacked air and water regulators. Therefore, we used transparent, punctured (pin holes) tygon tubing to generate the bubbles.

Once the hydrophobic particles enter the cleaning zone, most of the unwanted gangue is washed away by the wash water. A positive net downward bias results from this wash water flowrate. The froth zone permits the unwanted particles to drain back to the collection zone while retaining the hydrophobic minerals, which eventually discharge over the column lip.

Proof of this cleaning system being a much more feasible method is given by the Gaspé Mines (Québec) where two columns replaced 13 conventional cleaners in a molybdenum upgrading circuit with superior results. As well, the Magma Copper Company at San Manual, Arizona replaced a two-stage conventional cleaner cleaner circuit in copper recovery (grade 29.99% and recovery 90.12%) by a single-stage flotation column (grade 29.83% and recovery 90.36%).

Initial steps have been taken in commissioning the Department's 18 foot flotation column at the Dominion Explorers Inc. antimony mine in order to try to recover their fine particles. As well, preliminary steps have been taken to commission the flotation column at the Denison Co. Ltd. potash mines in Sussex, New Brunswick.

Discussion: 1.3 DESIGN VARIABLES

In order to go through the scale-up procedure with more ease, a flotation column simulator program is available from the University of Toronto (See Attachment A). The recovery calculations in the scale-up procedure are based on Figure 2, where particle collection is considered to be a first-order rate process. Once the collection zone and cleaning zone recoveries are known, the following equation is used.

$$R_i = \frac{Rc_i Rf_i}{(Rc_i Rf_i + 1 - Rc_i)}$$

In order to calculate the collection zone recovery, plug flow is assumed and the following equation is used.

$$R = 1 - \frac{4a \exp(\frac{1}{2}Nd)}{(1 + a^2)\exp(\frac{a}{2Nd}) - (1-a)^2 \exp(\frac{-a}{2Nd})}$$

where, $a = \sqrt{1 + 4K\gamma_p Nd}$

$$\gamma_p = \gamma_l \left(\frac{U_l}{U_l + U_p}\right) \qquad \gamma_p = \frac{Hc}{V_l}$$

$$U_p = \frac{g\ (\text{class size})^2\ (\mu m)}{18 \cdot \text{viscosity (poise)}}$$

$$U_L = \frac{\text{Feed Flowrate + Bias}}{\text{Area}}$$

$$E = 0.063\ Dc\ \frac{Qg}{1.6}$$

$$N_d = \frac{E}{(Hc\ (U_L + U_p))}$$

Author(s)	technical notes	Project No.	Page	of
C. Arseneau			5	6

Discussion:

If a flotation column is not well aligned, then a degree of misalignment must be included in the calculation. It may be calculated as follows:

$$\frac{E\alpha}{E} = (1 + 1100\ \ Dc)^2$$

As well, maximum loading in the collection zone is evaluated. By using the following equation, the maximum flux is calculated and later compared to the actual mass flux.

$$mc^1 = 6\,\gamma\ db^{-1}\ dp\ Pp\ Jg\ Ac \qquad \gamma = 0.8$$

If $mc^1 > mc$, the recoveries for the collection zone are recalculated using a smaller H_c value.

The cleaning zone recovery equation is based on measurements from a combination of counter-current and current laboratory tests. Froth zone recovery decreases with increasing particle size. The carrying capacity limitations are used to calculate the actual froth zone recovery as follows:

$$Ca = 0.068\ d_{80}\ P_p\ \text{(for } Jg > 1.5\ \text{cm/s)}$$

$$Ca = 0.068\ d_{80}\ \frac{Jg}{1.5}\ \frac{0.12}{db}\ \text{(for } Jg < 1.5\ \text{cm/s)}$$

Author(s)	technical notes	Project No.	Page	of
C. Arseneau			6	6

Discussion: 1.4 PROCEDURE

For best results, all 5 three-foot segments of the plexiglass column should be set up as near vertical as possible (Figure 1). Connected to the column is a peristaltic feed pump with a controller, a peristaltic wash water pump with a controller, as well as an air compressor with a flowmeter, pressure gauge, and pressure regulator.

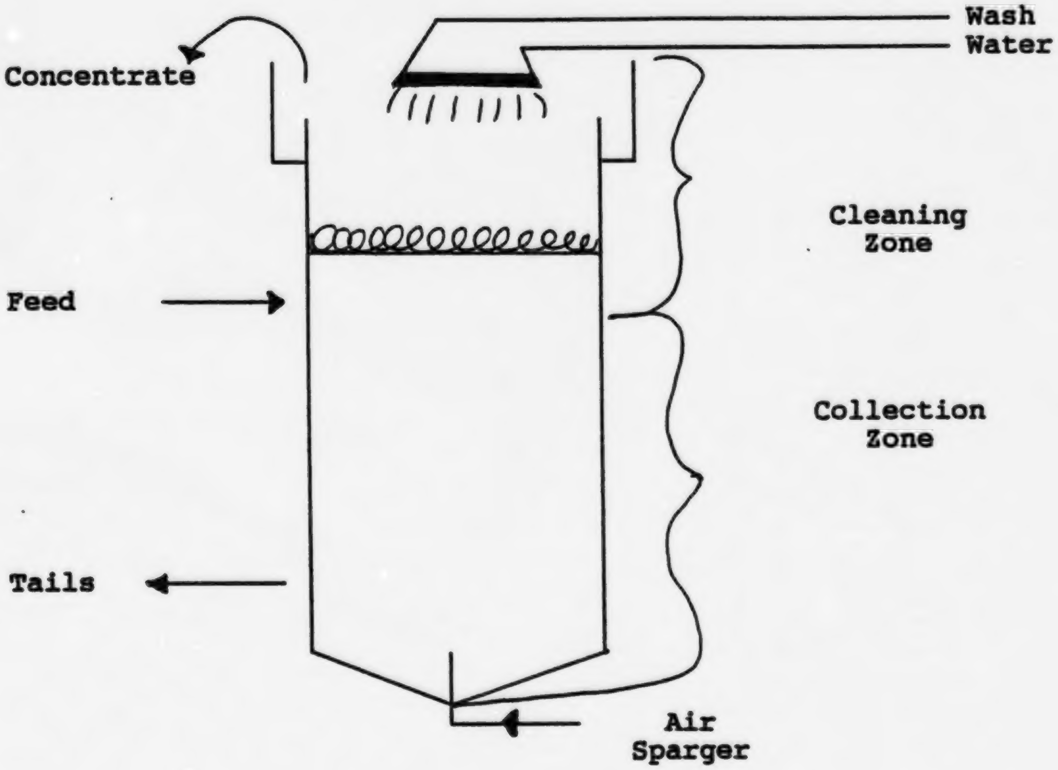
It is very important to run the column under stable conditions and at a steady state with only water and frother (MIBC). The mixture of all MIBC and water entering the column should be 3 mL of 1% MIBC for every litre of water.

To start up, the air compressor is turned on and air flow is adjusted to approximately 24 litres/minutes. The tailings pump should be shut off while the feed and wash water pump fill the column. An interface line is then visibly rising to the top. As the line approaches 2 to 3 feet from the top, the tailings pump is turned on in order to keep the interface at one level. The next step is to adjust the feed flowrate, wash water rate and tailings flow in order to get a positive downward bias flow of approximately 0.2 to 0.15. A recommended setting would be 649 mL/min for the feed, 250 mL/min for the wash water and 860 mL/min for the tailings (See Attachment A).

The column should be operating under two conditions for at least 30 minutes before any slurry is introduced to the feed stream. These conditions are (1) positive downward bias flow between 0.2 and 0.15; (2) static interface between 2 to 3 feet from the top.

The feed is then pumped into the column and samples are taken from the feed, concentrate and tailings. Flowrates and weight percentage of solids are also recorded.

FIGURE 1


Wash
Water
Concentrate
Cleaning
Zone
Feed
Collection
Zone
Tails
Air
Sparger

APPENDIX A

Users Manual for a Flotation Column Simulator

Software Developed
by

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April 1988

Reprinted with authorization of the author:

G. S. Dobby, P. Eng.

The author has requested that any further information on this system only be obtained by contacting the author personally, at the above noted address.

INTRODUCTION

The purpose of this manual is to describe the use of a steady state flotation column simulator (IBM-PC Fortran). The manual describes: (i) input information required for the simulator, (ii) a summary of the process execution, and (iii) a step by step description of the results. Also attached is a sample output from the simulator.

The objective of the simulator is to provide an aid in scale-up and design of flotation columns. A description of the model structure is given in the preprint (attached) "Fine Particle Separation Performance in a Flotation Column Under Conditions of Heavy Froth Loading", that describes use of the simulator in a study of zinc cleaning in a 2 m diameter column. The simulator does not attempt to quantify effects of operating variables such as froth depth and bias rate upon separation performance and capacity.

The simulator is for a single column; expansion to circuit simulation is currently being undertaken.

PROGRAM START-UP

The simulator is found on a 5.25" disk. After having activated the computer with a system (DOS) disk, insert the simulator disk and enter the command "dir" to produce a files directory. The output on the screen should show the name and content of each file below:

File name	Purpose
COFLOSI.FOR	Listing of program
COFLOSI.EXE	Execution of program
DEFAULT	Input Datafile
PART 1	Output file
PART 2	Output file
PART 3	Output file

To start the simulator the user types "COFLOSI". The computer will retrieve data from a previously created datafile "DEFAULT" and displays it on the screen for modification and data entry.

DATA ENTRY

In the first two lines the user responds by entering the material to be treated in the column and the flotation test number. Following this the input data is organized in three sections:

I) Mineral Characteristics
II) Feed Grade, Rate Constant and Froth recoveries
III) Column Characteristics

For the first section the following variables are entered:

1) Number of mineral species to a maximum of 5
2) Name of each mineral species
3) Number of size classes to a maximum of 7
4) Average particle diameter (used in the calculation of particle terminal velocity) and weight percent in each size class
5) Specific gravity of each mineral species

A summary of the entered data appears on the screen and the user will be asked if the above values are correct. At this point, one may correct any errors by typing "n", which causes the cursor to return to the first line of the data input, or "y" , to proceed.

In the second data entry section, the feed grade (%), rate constant (min^{-1}) and froth recovery (fraction) are entered for each mineral and for each size class (froth recovery is the fraction of particles that, having entered the froth zone on the surface of bubbles, reports to the froth overflow). After having entered the last value a summary of this section appears on the screen. The last lines show the grades of the minerals in the feed. It is important to note that the weight percent in each size class must add to 100%; if this is not the case then the user is warned, and corrections should be made at this point.

The third data entry section is for column characteristics:

1) Column diameter (m)
2) Number of sections in column (from axial baffling)
3) Collection zone length (m) [interface to sparger for large column, feed port to sparger for laboratory/pilot column]
4) Gas hold-up (fraction)
5) Bubble size at top of collection zone (cm)
6) Gas velocity, STP (cm/s)
7) Operating bias (cm/s) [typically 0.05 to 0.20 cm/s]
8) Weight percent solids expected in the concentrate

DATA ENTRY (continued)

9) d_{80} of concentrate (um) [used to calculate carrying capacity of the concentrate]
10) Density of solids in the concentrate (g/cm^3) [also used to calculate carrying capacity of the concentrate]
11) Fractional packing density [used to calculate solids loading of bubbles at the top of the collection zone]
12) Slope of maximum carrying capacity expression (usually 0.068), C_{am} = slope x d_{80} x solids density (g/min/cm^2) [d_{80} in μm]
13) Vertical misalignment (cm) [Vertical distance of misalignment over the collection zone length]
14) Solids feedrate (metric tonnes/hour)
15) Weight percent solids in the feed (%).

Finally, a summary of the data from the third section appears and, as before, corrections can be made if necessary.

PROCESS EXECUTION

Figure 1 shows a flowchart of the major modules and operations involved in the simulator. Once all the data is entered the computer program calls up "Reccoll "module to calculate the recovery in the collection zone on a size-by-size and mineral-by-mineral basis. Then the collection zone is checked to see if it is bubble load limited. If the calculated bubble load (B_{Lc}) is greater than the maximum bubble load (B_{Lm}), determined by estimating maximum bubble coverage, then the program calls on "Reccoll" module again to calculate the recovery in the collection zone, this time using a shorter collection zone length. It iterates on collection zone length until $B_{Lc}= B_{Lm}$. Otherwise, the program proceeds to calculate the total recovery, which is a function of the collection zone recovery and froth zone recovery (an input parameter); this is done by calling on the "Rectotal" module. Then the maximum carrying capacity C_{am} (C_{am} [g/min/cm2] = 0.068 x d_{80} [μm] x density of solids) and calculated carrying capacity C_{ac} are compared. If C_{ac} is greater than C_{am}, then the froth zone recoveries R_f are reduced and the program calls on "Rectotal" module again, iterating on R_f until C_{ac} is equal to C_{am}. Next, the column feed is combined with the dropback from the froth zone, whereupon the program returns to the first step. This is repeated until the subsequent difference between two circuit recoveries is less than 0.1 %, at which point the program is assumed to have reached steady-state.

DATA OUTPUT

During program execution, the progress of the simulation is shown on the screen by indication of the loop that the program is acting on. Once steady-state has been achieved, the program ends. Data output is divided into three parts:

1) Stream properties and mineral summary,
2) Input data summary, and
3) Size-by-size and mineral-by-mineral summary.

Part 1 of the output starts with a summary of the flotation column performance and displays (i) TPH of solids, (ii) weight % solids, (iii) solids, liquid and slurry flowrates, and (iv) slurry densities, for the feed, concentrate and tailings streams. Wash water addition and gas velocity are reported. Wash water addition has been determined by summing the bias flow water (an input) and water flow rate in the concentrate (calculated knowing concentrate solids rate and input values for solids density and concentrate slurry density). Next, the same stream properties as above are given for the collection and froth zones individually.

The second page of output commences with a mineral summary of the feed. For each mineral, the particle size, weight %, grade(%) and distribution (%) are shown for each size class. Next, a mineral summary of the concentrate is shown. First, the total weight recovery is shown followed by the recovery and grade of each mineral. Then in the same manner as above, for each mineral, the particle size, weight %, grade (%) and distribution (%) are shown for each size class of the concentrate.

In part 2 of the data output, a summary of the input data is reported. The effective collection zone length is also given; this is the length of the collection zone for which the bubbles are not fully loaded.

Part 3 shows a size-by-size and mineral-by-mineral summary. It starts with the effective liquid residence time and, for each mineral in each size class, particle residence time. The sequence presented is: mineral 1 - sizes 1,2,3, ...; mineral 2 - sizes 1,2, 3, ...; etc.

Next, the recoveries in the collection zone are shown. "Rcoll" represents the recovery of a mineral in each size class, "Rcmin" shows the net collection zone recovery of each mineral, "Rcsize" is the net collection zone recovery in each size class, and "Rctotal" is the total recovery in the collection zone.

The next block displays the recoveries in the froth zone. First shown are the actual froth zone recoveries which will be different from the input values if the column is carrying capacity limited. "Rclmin" represents the recovery of each mineral in the froth zone and "Rcltot" is the total recovery in the froth zone.

DATA OUTPUT (continued)

The overall column recoveries follows, with "Rt" representing the recovery of a mineral in each size class. "Rtmin" is the recovery of each mineral. "Rttotal" is the total recovery to the column.

The internal circulating loads (ICL) within the collection and froth zones have been calculated. "CL" represents the ICL of a mineral in each size class. "CLmin" is the ICL of each mineral. "CLtotal" is the total ICL between the two zones.

Finally, data for the following streams is displayed

- Column feed
- Collection zone feed
- Collection zone tailings (i.e. column tailings)
- Cleaning zone concentrate
- Cleaning zone tailings

For each of these streams results are given for:

i) Solids distribution to each size class
ii) Grade of mineral in each size class
iii) Overall grade of mineral
iv) Distribution of mineral to each size class.

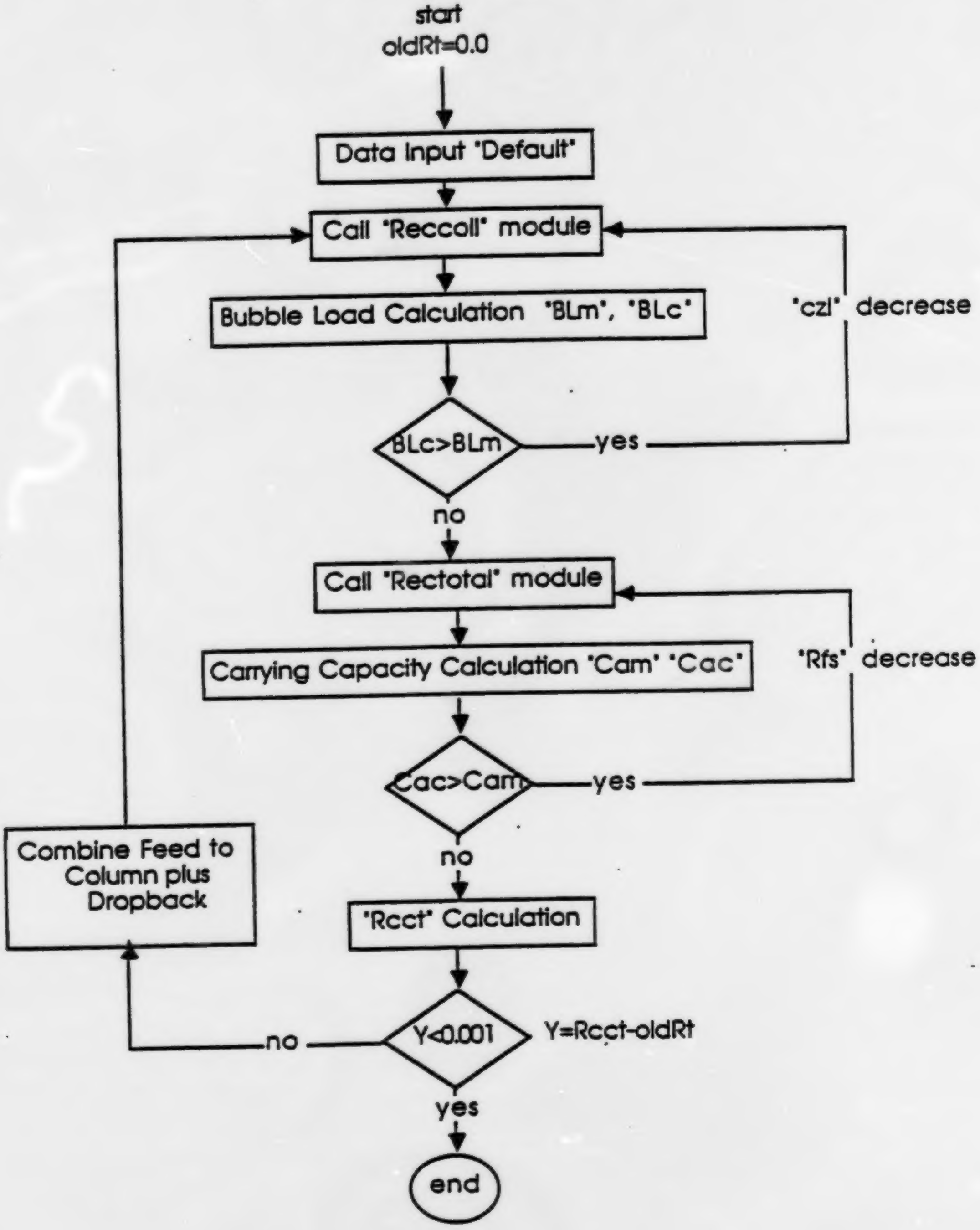
start
oldRt=0.0
Data Input 'Default'
Call 'Reccoll' module
Bubble Load Calculation 'BLm', 'BLc'
'czl' decrease
BLc>BLm
yes
no
Call 'Rectotal' module
Carrying Capacity Calculation 'Cam' 'Cac'
'Rfs' decrease
Cac>Cam
yes
no
Combine Feed to Column plus Dropback
'Rcct' Calculation
Y<0.001
Y=Rcct-oldRt
no
yes
end

Figure 1 : Calculation Flowchart For COFLOSI

```
                              COLUMN FLOTATION SIMULATION
                              ***************************

For material:Zn Rougher Conc                                  Test #:225


        THE CIRCUIT REACHED STEADY STATE AFTER   9 ITERATIONS.


     STREAM PROPERTIES :                      COLUMN SUMMARY
     ===================

                               FEED       CONC      TAILS
          TPH SOLIDS :         20.00      11.46      8.54
 WT  PERCENT SOLIDS :          35.00      35.00     14.98
 SOLIDS FLOW(L/min) :          81.56      44.05     37.50
 LIQUID FLOW(L/min) :         619.05     354.81    807.54
 SLURRY FLOW(L/min) :         700.60     398.86    845.04
     SLURRY DENSITY :           1.36       1.37      1.12

WASH WATER ADDITION (L/MIN) =  543.30
       GAS VELOCITY (CM/S) =    2.00

                                      COLLECTION ZONE
                               FEED       CONC      TAILS
          TPH SOLIDS :         26.40      17.86      8.54
 WT  PERCENT SOLIDS :          35.27     100.00     14.98
 SOLIDS FLOW(L/min) :         106.13      68.63     37.50
 LIQUID FLOW(L/min) :         807.54        .00    807.54
 SLURRY FLOW(L/min) :         913.67      68.63    845.04
     SLURRY DENSITY :           1.         4.34      1.12

                                      CLEANING ZONE
                               FEED       CONC      TAILS
          TPH SOLIDS :        -17.86      11.46      6.40
 WT  PERCENT SOLIDS :         100.00      35.00     36.14
 SOLIDS FLOW(L/min) :          68.63      44.05     24.58
 LIQUID FLOW(L/min) :         543.30     354.81    188.50
 SLURRY FLOW(L/min) :          68.63     398.86    213.07
     SLURRY DENSITY :           4.34       1.37      1.39



                                                              pg 2.


                             MINERAL SUMMARY
                             ===============

          FEED:
```

```
          (um)      (%)        (%)        (%)
 1        4.00     20.00      41.00      20.00
 2        8.00     20.00      41.00      20.00
 3       12.00     20.00      41.00      20.00
 4       20.00     20.00      41.00      20.00
 5       30.00     20.00      41.00      20.00
 Pyrite
Size      dp        Wt.       grade      distn
          (um)      (%)        (%)        (%)
 1        4.00     20.00      39.00      20.00
 2        8.00     20.00      39.00      20.00
 3       12.00     20.00      39.00      20.00
 4       20.00     20.00      39.00      20.00
 5       30.00     20.00      39.00      20.00
 Gangue
Size      dp        Wt.       grade      distn
          (um)      (%)        (%)        (%)
 1        4.00     20.00      20.00      20.00
 2        8.00     20.00      20.00      20.00
 3       12.00     20.00      20.00      20.00
 4       20.00     20.00      20.00      20.00
 5       30.00     20.00      20.00      20.00

CONCENTRATE:    Total Wt. Recovery is 57.32 %

MINERAL               RECOVERY   GRADE
                        (%)       (%)

 Sphalerite            90.02     60.35
 Pyrite                62.44     36.58
 Gangue                11.02      3.07


 Sphalerite
Size      dp        Wt.       grade      distn
          (um)      (%)        (%)        (%)
 1        4.00      9.82      71.72      11.67
 2        8.00     18.37      63.72      19.39
 3       12.00     22.29      59.72      22.06
 4       20.00     25.01      56.23      23.31
 5       30.00     24.51      58.03      23.57
 Pyrite
Size      dp        Wt.       grade      distn
          (um)      (%)        (%)        (%)
 1        4.00      9.82      26.18       7.03
 2        8.00     18.37      34.21      17.18
 3       12.00     22.29      37.21      22.67
 4       20.00     25.01      40.30      27.56
 5       30.00     24.51      38.15      25.56
 Gangue
Size      dp        Wt.       grade      distn
          (um)      (%)        (%)        (%)
 1        4.00      9.82       2.10       6.70
 2        8.00     18.37       2.07      12.37
 3       12.00     22.29       3.07      22.29
 4       20.00     25.01       3.47      28.20
 5       30.00     24.51       3.82      30.44
```

```
                                   PART 2

                                   pg 1

                SUMMARY OF INPUT DATA
                =====================


Number of Mineral Species        : 3
     Name of Species # 1         : Sphalerite
     Name of Species # 2         : Pyrite
     Name of Species # 3         : Gangue
Number of particle class sizes   :    5
Average diameter of class # 1    :    4.0 microns
And wt % of total solids in class:   20.00
Average diameter of class # 2    :    8.0 microns
And wt % of total solids in class:   20.00
Average diameter of class # 3    :   12.0 microns
And wt % of total solids in class:   20.00
Average diameter of class # 4    :   20.0 microns
And wt % of total solids in class:   20.00
Average diameter of class # 5    :   30.0 microns
And wt % of total solids in class:   20.00
Spec. gravity of Sphalerite      :   4.100
Spec. gravity of Pyrite          :   5.000
Spec. gravity of Gangue          :   3.000


                 COLUMN DATA
                            Column
            Diameter(m):     2.000
     Number of sections:         1
     Baffled Diameter(m):    2.000
 Collect. Zone Length(m):   14.000
 Effective CZ Length (m):   14.000
 Gas Hold-up  (fraction):     .191
   Ave. Bubble Size (cm):      120
     (STP) Gas Vel(cm/s):    2.000
   Operating Bias (cm/s):     .100
 Wt % solids in the conc:   35.000
 80% passing size to con:   25.000
     Density of solids  :    4.300
 Fract. packing density :     .800
           Slope of Cam:      .060
          Misalignment  :     .000
    Angle of inclination:     .000
```

FEED GRADE, RATE CONSTANT AND FROTH RECOVERIES

Sphalerite

Size	Grade (%)	Rate Constant (min-1)	Input Rfs (%)
1	41.00	.0290	78.00
2	41.00	.1100	74.00
3	41.00	.2400	70.00
4	41.00	.5600	62.00
5	41.00	.3500	53.00

Pyrite

Size	Grade (%)	Rate Constant (min-1)	Input Rfs (%)
1	39.00	.0075	78.00
2	39.00	.0275	74.00
3	39.00	.0500	70.00
4	39.00	.0950	62.00
5	39.00	.0950	53.00

Gangue

Size	Grade (%)	Rate Constant (min-1)	Input Rfs (%)
1	20.00	.0010	78.00
2	20.00	.0020	74.00
3	20.00	.0040	70.00
4	20.00	.0060	62.00
5	20.00	.0080	53.00

```
                         SIZE-BY-SIZE AND MINERAL SUMMARY
                         ================================

Effective Liquid and Particle Residence Time (min)
 Liq. res. time        38.94
part. res. time        38.83    38.50    37.97    36.35    33.56    38.80    38.36    37.66    35.57    32.10    38.88
38.68    38.36    37.35    35.54

 Collection Zone Recoveries (%)
 Rcoll                 55.43    85.86    95.02    98.93    99.70    23.00    53.68    69.03    82.17    80.55     3.75
 7.23     13.46    18.62    22.61
 Rcmin                 89.31    64.64    13.39
 Rcsize                33.43    60.03    71.64    80.33    81.66
 Rctotal               67.67

 Froth Zone Recoveries (%)
 Actual Rfs            78.00    74.00    70.00    62.00    53.00    78.00    74.00    70.00    62.00    53.00    78.00
74.00    70.00    62.00    53.00
 Rclmin                64.62    63.58    62.47
 Rcltot                64.17

 Overall Column Recoveries (%)
 Rt                    49.24    81.80    93.04    98.32    99.43    18.89    46.17    60.94    74.08    68.70     2.95
 5.45     9.82     12.42    13.41
 Rtmin                 90.02    62.44    11.02
 Rttotal               57.32

 Internal Circulating Loads (%)
 CL                    13.89    28.74    39.87    60.26    88.17     5.23    16.22    26.12    45.40    60.92      .83
  .91     4.21     7.61     11.83
 CLmin                 49.28    35.77     6.62
 CLtotal               43.19

 Solids Distribution to Size Classes (%)
 Column Feed           20.00    20.00    20.00    20.00    20.00
 Collection Feed       16.35    17.95    19.30    21.81    24.59
 Collection Conc.       8.08    15.93    20.43    25.89    29.67
 Collection Tails      33.67    22.19    16.93    13.27    13.94
 Cleaning Con.          9.82    18.37    22.29    25.01    24.51
 Cleaning Tails         4.96    11.56    17.11    27.46    38.92
```

```
 Grade of Mineral in each Size Class (%)                                   39.00    39.00    39.00    39.00    39.00    20.00
 Column Feed           41.00    41.00    41.00    41.00    41.00
-- --    -- --    -- --    -- --
```

```
 Collection Feed      43.26     44.55     45.02     45.65     47.53     38.06     38.25     38.61     39.40     38.67     18.68
17.20     16.36     14.95     13.79
 Collection Conc.     71.72     63.72     59.72     56.23     58.03     26.18     34.21     37.21     40.30     38.15      2.10
 2.07      3.07      3.47      3.82
 Collection Tails     28.96     15.75      7.90      2.43       .78     44.02     44.32     42.17     35.71     41.02     27.01
39.32     49.93     61.86     58.19
 Cleaning Con.        71.72     63.72     59.72     56.23     58.03     26.18     34.21     37.21     40.30     38.15      2.10
 2.07      3.07      3.47      3.82
 Cleaning Tails       71.72     63.72     59.72     56.23     58.03     26.18     34.21     37.21     40.30     38.15      2.10
 2.07      3.07      3.47      3.82

 Grade of mineral (%)
 Column Feed          41.00     39.00     20.00
 Collection Feed      45.40     38.64     15.95
 Collection Conc.     59.92     36.92      3.16
 Collection Tails     15.02     42.25     42.73
 Cleaning Conc.       60.35     36.58      3.07
 Cleaning Tails       59.16     37.53      3.31

 Distribution of Mineral in each Size Class (%)
 Column Feed          20.00     20.00     20.00     20.00     20.00     20.00     20.00     20.00     20.00     20.00     20.00
20.00     20.00     20.00     20.00
 Collection Feed      15.58     17.61     19.14     21.93     25.74     16.11     17.77     19.28     22.23     24.61     19.15
19.36     19.79     20.44     21.25
 Collection Conc.      9.67     16.94     20.36     24.30     28.73      5.73     14.76     20.59     28.26     30.66      5.37
10.44     19.89     28.41     35.88
 Collection Tails     64.94     23.28      8.90      2.15       .73     35.08     23.28     16.89     11.21     13.54     21.28
20.74     19.78     19.21     18.99
 Cleaning Conc.       11.67     19.39     22.06     23.31     23.57      7.03     17.18     22.67     27.56     25.56      6.70
12.37     22.29     28.20     30.44
 Cleaning Tails        6.01     12.45     17.27     26.10     39.18      3.46     10.53     16.96     29.48     39.56      3.15
 7.24     15.90     28.77     44.94
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Fine Particle Separation Performance in a Flotation Column Under Conditions of Heavy Froth Loading

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ABSTRACT

Many applications of column flotation operate at a high level of solids loading in the froth; for example this is often the case for column cleaning of fine (< 30 µm) sulfides. Under these conditions the performance of the froth zone can strongly affect the unit capacity and the separation efficiency. This paper describes a steady-state computer simulation method for assessing column performance under such conditions and presents some of the findings. The effects of feed rate, slurry density, axial baffling, bubble size, and gas rate are described.

KEYWORDS

bubbles, flotation columns, froths, gas holdup, separation, kinetics, simulation

INTRODUCTION

The benefits of using flotation columns are maximized when applied to the cleaning of fine grained minerals and/or when a large degree of upgrading is required, eg. molybdenite concentration. When fine grained material is subjected to cleaning it will often be the case that the limiting capacity of a column is determined by the solids carrying capacity of the froth. This paper examines some of the features of column flotation under conditions of high solids loading within the froth.

The assessment of the effects of flotation parameters is difficult to achieve experimentally. This is due to the complex nature of the feed material and to the internal interaction between the froth phase and the pulp, in the form of two-way transfer of solids and liquid; this interaction is unmeasureable through conventional flotation testing. The purpose of flotation simulators (mathematical models) is to make parameter assessment in a controlled manner. Of course, the necessary ingredient for successful simulator development is a good understanding of the fundamental microprocesses that are occurring. Column flotation is amenable to this type of simulation. Relative to mechanical cell flotation, a few features are somewhat simplified: (i) the flow regime in the pulp, usually not having a mechanical mixing component, can be reliably described, (ii) recovery by physical entrainment can be considered to be negligible, and (iii) the column froth consists of a single three-dimensional phase, whereas in a bank of mechanical machines the froth properties often vary along the length of the bank.

Using the information obtained from column research during the past several years, a steady-state flotation column simulator has been developed. The simulator uses size-by-size and mineral-by-mineral rate data, and accounts for solids overloading in the froth, as well as collection zone overloading. It complements a column circuit simulator (Yianatos and colleagues, 1987) that uses a less complicated column model.

The system examined through the use of the simulator and described here is a fine grained (80% < 25 µm) zinc sulfide rougher concentrate consisting of sphalerite, pyrite and non-sulfide gangue.

SIMULATOR DESCRIPTION

The basic structure of the column simulator has been described before (Dobby and Finch, 1986). Particle collection is considered to be a first-order rate process, on a size-by-size and mineral-by-mineral basis. Collection zone recovery for a given particle size class and mineral, R_{Ci}, is calculated using the plug-flow dispersion model (Levenspiel, 1972). Froth zone recovery, R_{Fi}, is the recovery to the froth overflow of particles entering the froth from the collection zone attached to gas bubbles. Recovery by entrainment is assumed to be zero. Then, overall recovery, R_i, to the froth overflow from the column feed is given by

$$R_i = R_{Ci} \cdot R_{Fi} / (R_{Ci} \cdot R_{Fi} + 1 - R_{Ci}) \qquad (1)$$

Collection Zone Recovery

The vessel dispersion number, N_d, is the parameter that describes the mixing conditions in the collection zone:

$$N_d = E / [H_C (u_l + u_p)] \tag{2}$$

where E is the dispersion coefficient, H_C is the collection zone length, and u_l and u_p are liquid interstitial velocity and particle settling velocity, respectively. E is proportional to the column diameter (Dobby and Finch, 1985), the equivalent diameter of an individual section in the case of a baffled column. Non-vertical alignment of a bubble column has been shown (Tinge and Drinkenburg, 1986) to have a very significant impact upon the dispersion coefficient. Letting E_α be the dipersion coefficient when the column is tilted at an angle α to the vertical, then

$$E_\alpha / E = (1 + 1100\, \alpha\, D_C)^2 \tag{3}$$

where α is in radians and D_C is in metres; this correlation was found for columns up to $D_C = 0.2$ m and H_C/D_C as low as 15/1. Equation (3) is used for this study; however, verification at larger diameters is warrented. For $D_C = 1$ m, a horizontal misalignment of 1 cm over a 10 m height gives $E_\alpha/E = 4.4$.

Particle retention time in the collection zone, τ_p, is given by

$$\tau_p = \tau_l [\, u_l / (u_l + u_p)\,] \tag{4}$$

where τ_l is liquid retention time. u_p is calculated using Masliyah's (1979) equation for hindered settling in multi-species particle systems.

Bubble overloading in the collection zone is evaluated by comparing the mass flux of solids calculated to be entering the froth zone from the collection zone, m_C, with the maximum mass flux obtainable, $\dot{m}_C'$. The latter is given by assuming a monolayer coverage of particles at the interfacial area available at the top of the collection zone:

$$\dot{m}_C' = 6\, \gamma\, d_b^{-1}\, d_p\, \rho_p\, J_g\, A_C \tag{5}$$

where γ is packing density of solids on the surface of a bubble (0.8 in this work), d_b is bubble diameter, d_p is particle diameter (for a size distribution d_{65} is usually suitable), ρ_p is particle density, J_g is superficial gas velocity, and A_C is column cross-sectional area. If $m_C > m_C'$ then collection recoveries are recalculated using a smaller value of H_C, and recalculation iterates on H_C until $m_C = m_C'$. The physical situation is a top section of the collection zone where no mass transfer of particles from the liquid to the bubbles can occur, i.e. a nonfunctional volume.

Froth Zone Recovery

Froth recoveries have been measured using a combination of countercurrent and cocurrent laboratory column operation (Contini and colleagues, 1988). Froth zone recovery decreases for increasing particle size; for example, using a silica system that was not solids carrying limited in the froth, R_f decreased from 68 % for -14 µm particles to 34 % for 40 µm particles. The same relative variation in R_f is used as the input values in this study (the actual R_f values are determined by carrying capacity limitations).

Using the same method, froth recoveries for a sphalerite-pyrite system have been determined (again when the froth was not solids carrying limited). The difference between R_f of sphalerite and R_f of pyrite was insignificant; R_f for sphalerite was 52% and R_f for pyrite was 53% for particles between 10 and 37 µm. Therefore, this study assumed that R_f is independent of mineral type. [Measurements of froth upgrading, from the interface to the top of the froth, have shown both the existence of some upgrading in a molybdenite cleaner (Yianatos, 1987) and no upgrading in a copper cleaner (Amelunxen and colleagues, 1988).]

Maximum carrying capacity, C_a (in terms of mass flux of concentrate solids per unit area), has been shown empirically (Espinosa-Gomez and colleagues, 1988b) to be given by

$$C_a = 0.068\, d_{80}\, \rho_p \qquad [\text{for } J_g > 1.5 \text{ cm/s}] \qquad (6a)$$

where C_a is in g/min/cm^2, d_{80} is the 80% passing size of the solids, in µm, and ρ_p is in g/cm^3. C_a is independent of gas rate at $J_g > 1.5$ cm/s (Espinosa-Gomez and colleagues, 1988a). At lower gas rates, C_a is a function of J_g and d_b; it is suggested that

$$C_a = 0.068\, d_{80}\, \rho_p\, (J_g/1.5)\,(0.12/d_b) \qquad [\text{for } J_g < 1.5 \text{ cm/s}] \qquad (6b)$$

where d_b is in cm. Some experimental evidence of this relationship has been obtained and is to be reported elsewhere.

When the calculated concentrate solids flux exceeds $C_a \cdot A_c$, calculations are repeated at lower R_f values (R_f is changed proportionately for all size classes), and recalculation iterates on R_f until the calculated concentrate solids flux equals $C_a \cdot A_c$.

CONDITIONS FOR CASE STUDY

All simulations reported here are for a 2 m diameter column with $H_C = 10$ m. The bias flow of water through the froth is at a superficial rate of 0.1 cm/s (188 L/min).

Feed

The column feed simulates a zinc rougher concentrate consisting of sphalerite (41%), pyrite (39%) and non-sulfide gangue (20%). The sphalerite is 63% Zn. Feed particle size distribution is: $d_{80} = 25$ µm, $d_{50} = 10$ µm, and $d_{20} = 6$ µm.

Feed is at 25, 35 or 45 weight % solids, and feed solids rate is from 20 to 80 tonnes/hour. At 35% solids the slurry superficial velocity, J_{sl}, is 0.47 cm/s for 20 TPH feed and 1.58 cm/s for 80 TPH. At 25% solids J_{sl} is from 0.67 to 2.39 cm/s, and at 45% solids J_{sl} is from 0.36 to 1.13 cm/s.

Collection rate constants have been simulated by first selecting reasonable values for (a) d_{opt}, the particle size at which collection is maximized, and (b) k_{opt}, the rate constant at d_{opt}, and then applying a model of size-by-size particle collection (Wilson, 1987). For a given particle size, collection rate constant k is given by

$$k = b\, d_p^{\,2} \exp[-1.14\,(d_p/d_{opt})^{1.76}] \qquad (7)$$

where b is determined knowing d_{opt} and k_{opt}. This model has been shown to describe the size-by-size distribution of collection rate constants for systems of silica, galena and hematite. The following values have been used for the base case in this study: sphalerite – $d_{opt} = 45$ µm and $k_{opt} = 1.2$ min^{-1}; pyrite – $d_{opt} = 25$ µm and $k_{opt} = 0.2$ min^{-1}. (Arbitrary, and low, k values for non-sulfide gangue were used, to reflect some degree of particle interlocking.) The base case is at $J_g = 2$ cm/s (STP) and $d_b = 0.12$ cm at the top of the collection zone.

To simulate bubble sizes different from the base case (0.12 cm), the base case rate constants were multiplied by the factor $(d_b/0.12)^2$, stating essentially that rate constant is proportional to d_b^{-2}. This is a sufficient estimator for these purposes (Dobby and Finch, 1987).

To simulate gas velocities different from the base case (2 cm/s), it was assumed that 2 cm/s was the gas velocity at which the collection rate constant was maximum. Varying the gas rate by ±0.6 cm/s (on either side of 2 cm/s) will have relatively little effect upon k (Dobby and Finch, 1986). Over the range 1.3 to 2.6 cm/s it is assumed that k is constant.

Initial values for froth zone recoveries were:

d_p(µm):	4	6	12	20	30
R_F (%):	78	74	70	62	53

Final froth zone recoveries were determined by the feed rate and the carrying capacity limitation (discussed later).

Gas Holdup

Gas holdup, ϕ_g, in the collection zone is a function of several variables:

$$\phi_g = fn\,(d_b, \rho_b, J_g, J_{sl}, \rho_{sl}, \mu_{sl})$$

where ρ_b is bubble-particle aggregate density, and ρ_{sl} and μ_{sl} are slurry density and viscosity respectively. Maximum bubble density is obtained by considering the monolayer coverage of particles on a bubble. This gives

$$\rho_{bmax} = d_p \rho_p \gamma \,/\, (d_b/6 + d_p \gamma) \qquad (8)$$

where ρ_p is particle density. In this case (high solids feed rates to the collection zone) it is reasonable to assume that the average ρ_b is about 2/3 of ρ_{bmax}. The table below summarizes the range of bubble size and bubble densities used.

d_b-STP (cm):	0.08	0.10	0.12	0.15	0.18
d_b- average:	0.07	0.083	0.105	0.13	0.16
ρ_b (g/cm^3):	0.34	0.28	0.23	0.19	0.16

The relationship between gas holdup and these operating variables has been described elsewhere (Dobby and colleagues, 1988; Yianatos and colleagues, 1988). Figure 1(a) shows calculated gas holdup versus gas velocity for several bubble sizes at a feed rate of 20 TPH and 35% solids (J_{sl} = 0.47 cm/s). Figure 1(b) is the same data presented for a higher slurry velocity (60 TPH and 25% solids; J_{sl} = 1.82 cm/s). Clearly, as bubble size decreases gas holdup increases (for constant J_g). Perhaps the most important feature to note from Figure 1 is that there is a minimum bubble size that can be accepted for given conditions of J_g and J_{sl}. For example, at J_g = 1.5 cm/s (STP) the minimum bubble diameter is about 0.10 cm for J_{sl} = 0.47 cm/s, and about 0.12 cm for J_{sl} = 1.82 cm/s. At smaller bubble sizes the system is not in a bubbly flow regime, flooding of the column occurs, and the column is inoperable (Dobby and Finch, 1986a).

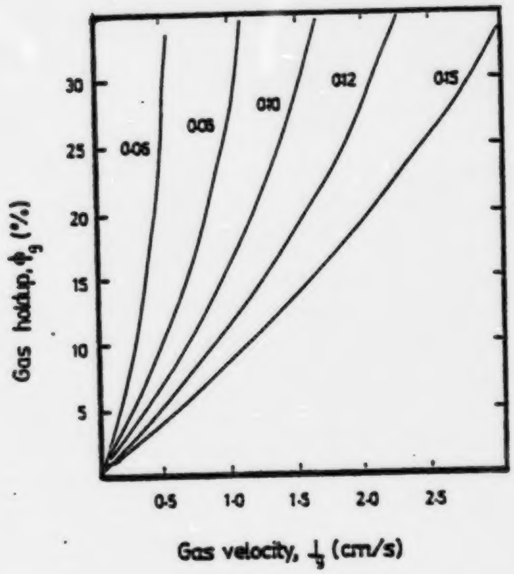

Figure 1(a).
Gas holdup vs gas velocity.
Numbers refer to d_b (cm).
Feed: 20 TPH 35% solids
J_{sl}=0.47 cm/s

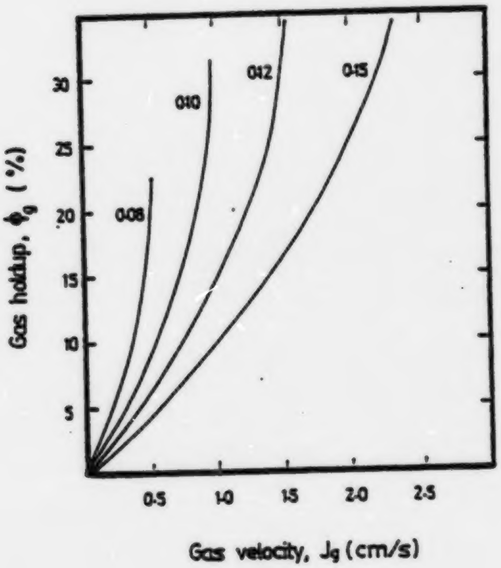

Figure 1(b).
Gas holdup vs gas velocity.
Numbers refer to d_b (cm).
Feed: 60 TPH 25% solids
J_{sl}=1.82 cm/s

CASE STUDY SIMULATION RESULTS

In almost all cases examined the froth zone is carrying capacity limited. An example of froth zone recoveries as a function of particle size is shown in Figure 2 for feed rates between 20 and 80 TPH. At feed rates greater than 40 TPH the internal circulating load of solids (from the froth to the pulp) is 300 to 500%.

The bubbles in the collection zone do not become solids-saturated for most conditions. The results of a case when the bubbles do become fully loaded before reaching the collection zone are shown in Figure 3. Effective H_c is the collection zone length over which bubbles are not fully loaded with solids. In the example of Figure 3 solids saturation of the bubbles occurs because gas-liquid interfacial area is quite low - J_g is small and d_b is large.

The remaining data presentation is in the form of Zn grade-recovery curves. Figure 4 shows that pulp density does not significantly affect the grade-recovery relationship. Also, there is little impact upon unit capacity for pulp densities greater than 35% solids. In systems that are not froth carrying capacity limited pulp density impacts upon retention time and therefore affects the capacity (or recovery). (The simulation has not allowed for a pulp density effect upon rate constants; experimental quantification of this effect is difficult and has not been reported.)

Axial baffles are expected to improve column performance because they reduce the effective diameter and increase the effective H_c/d_c ratio. The effect of baffling in this application is summarized in Figure 5. Four situations are presented:
(i) no baffles and vertical ($\alpha=0$),
(ii) baffled to four sections and both column and baffles are vertical ($\alpha=0$),
(iii) no baffles and vertically misaligned by 1 cm in 10 m ($\alpha=0.001$), and
(iv) baffled to four sections and vertically misaligned ($\alpha=0.001$).
Comparing (i) and (iii), or (ii) and (iv), shows that misalignment of the column and/or the baffles can have a large detrimental effect upon column performance. Non-vertical baffles may actually decrease the performance of an otherwise vertical column, rather than providing the expected improvement.

While the relationship developed by Tinge and Drinkenburg (equation (3)) has been extrapolated for application to large columns, there is little doubt that poorly installed columns will not perform to their potential. However, comparison of (i) and (ii) in Figure 4 shows that truely vertical baffling will improve the separability. This last statement assumes that feed and gas are properly distributed to all four sections, not necessarily a trivial accomplishment.

Bubble size effects are summarized in Figure 6 for $J_g = 2$ cm/s. Separation performance

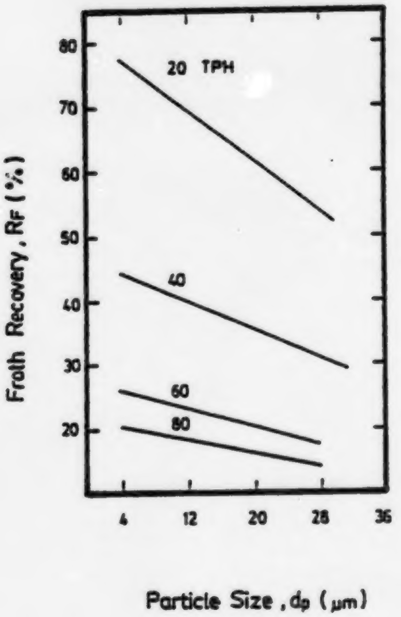

Figure 2.
Froth recovery vs particle diameter.
J_g = 2 cm/s; d_b = 0.12 cm;
35% solids; no baffles

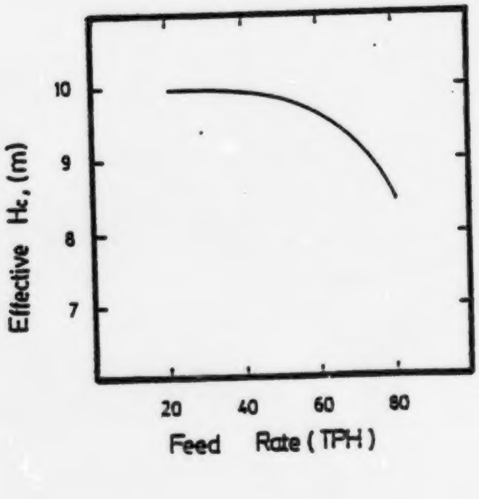

Figure 3.
Effective H_c vs feed rate.
J_g = 1.3 cm/s; d_b = 0.15 cm;
35% solids; no baffles

is improved when smaller bubbles are generated. However, this improvement is significant only when relatively large bubbles are reduced in size, eg. 0.18 cm reduced to 0.15 cm, or 0.15 cm reduced to 0.12 cm. Recalling Figure 1, at 2 cm/s a bubble diameter of about 0.12 cm is the limiting minimum size at the higher slurry flows.

Lastly, gas rate has been examined. In these simulations, the grade-recovery relationship is virtually identical at gas rates between 1.3 and 2.6 cm/s.

CONCLUSIONS

In column systems that are fed at a high rate with relatively fine, hydrophobic solids the following observations are expected:

- minimum bubble size (at the top of the collection zone) is about 0.1 to 0.12 cm,
- a bubble size substantially larger than this minimum will degrade the separability,
- slurry density does not significantly affect the grade-recovery relationship, and
- vertical baffling can improve separability, but only if the baffles are indeed vertical.

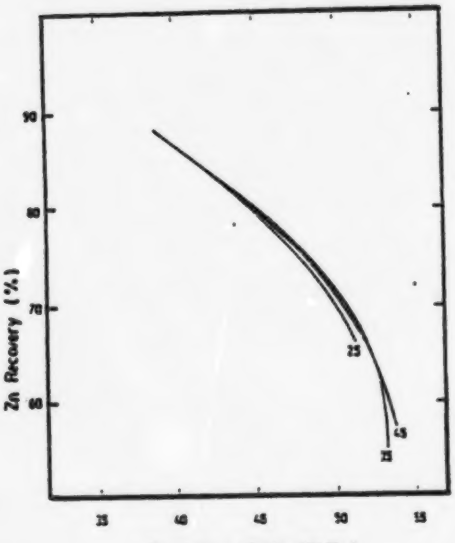

Figure 4.
Effect of pulp density on
Zn grade-recovery.
Numbers refer to wt % solids.
$J_g = 2$ cm/s; $d_b = 0.12$ cm;
no baffles

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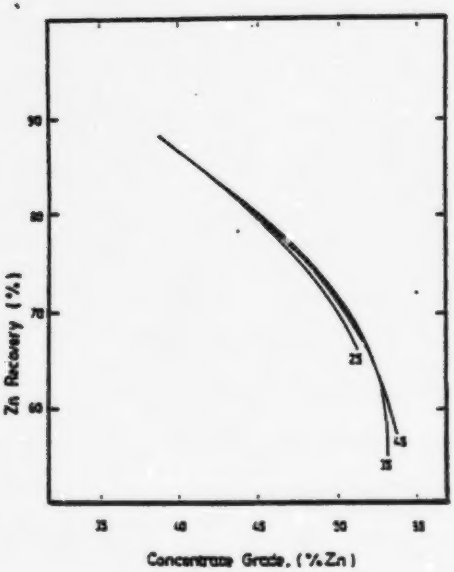

Figure 4.
Effect of pulp density on
Zn grade-recovery.
Numbers refer to wt % solids.
J_g = 2 cm/s; d_b = 0.12 cm;
no baffles

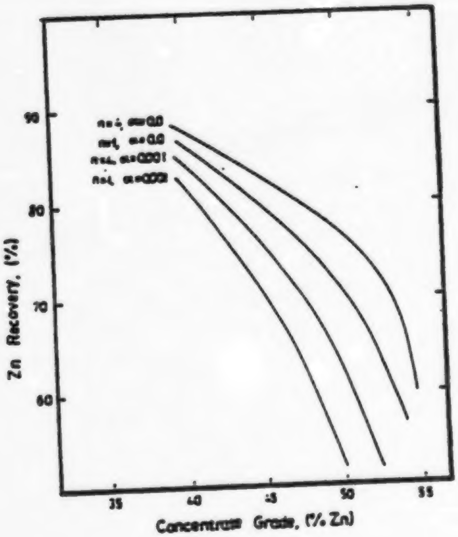

Figure 5.
Effect of baffling and misalignment on Zn grade-recovery.
n is the number of baffled sections.
J_g = 2 cm/s; d_b = 0.12 cm; 35% solids

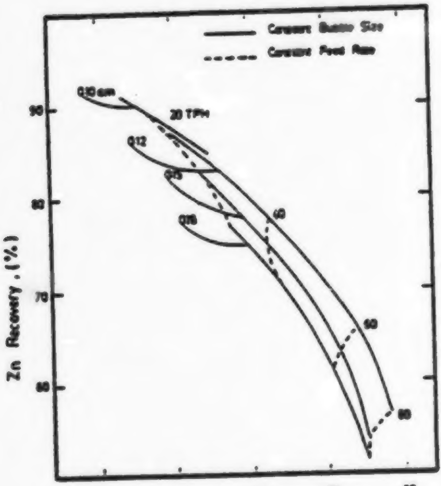

Figure 6.
Effect of bubble size on Zn grade-recovery
J_g = 2 cm/s; 35% solids;
no baffles

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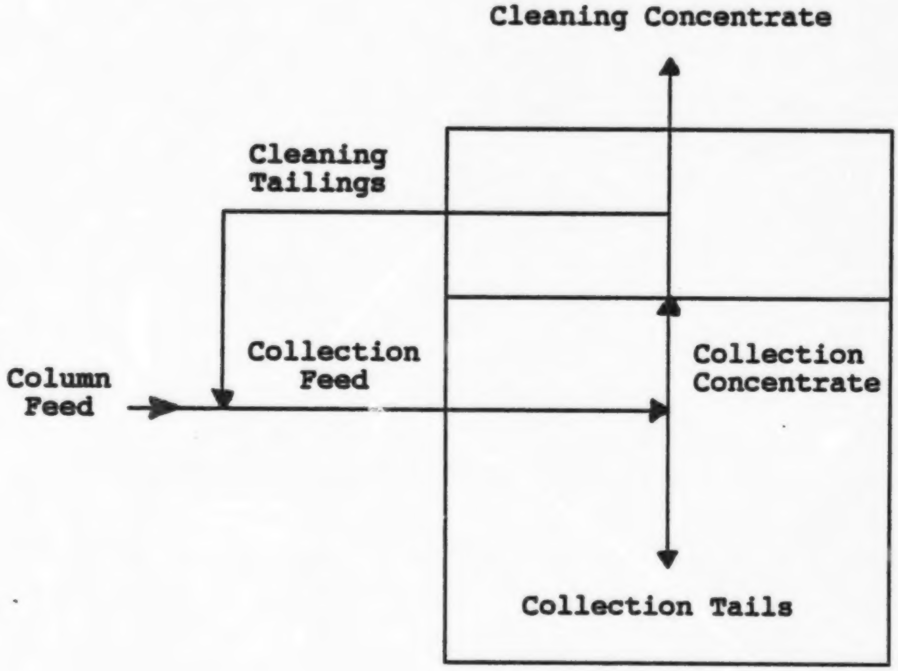
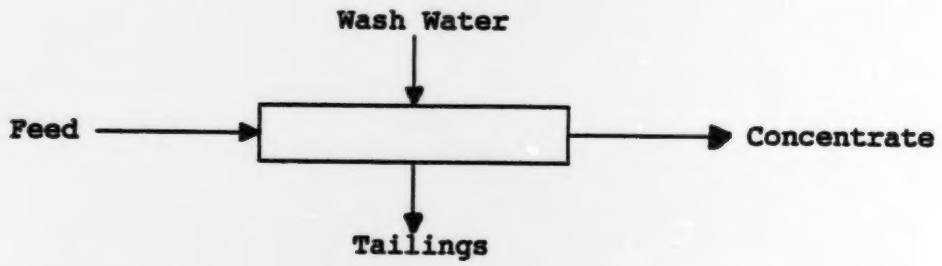
FIGURE 2

FLOWSTREAMS OF SIMULATOR MASS BALANCE PROGRAM


Cleaning Concentrate
Cleaning Tailings
Collection Feed
Column Feed
Collection Concentrate
Collection Tails

CALCULATION FOR BIAS FLOWRATES


Wash Water
Feed
Concentrate
Tailings

Feed + Wash Water = Tailings + Concentrate

Wash Water - Water Concentrate = Bias = Water Tailings + Water Feed